number 2

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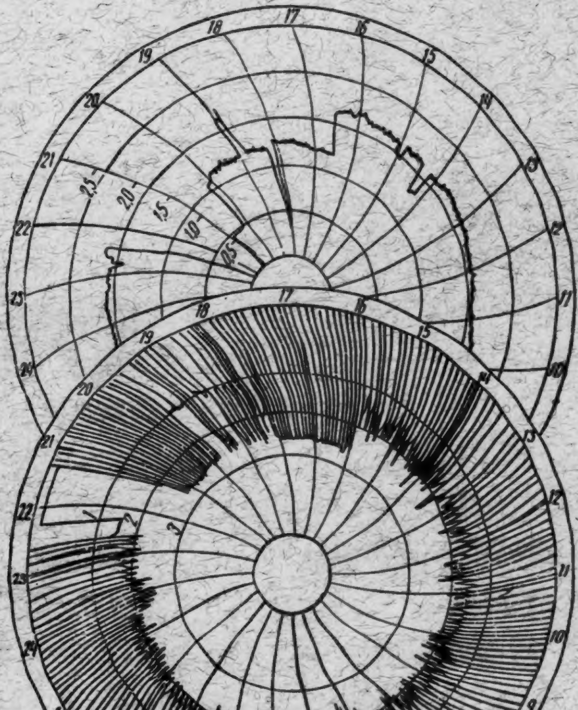

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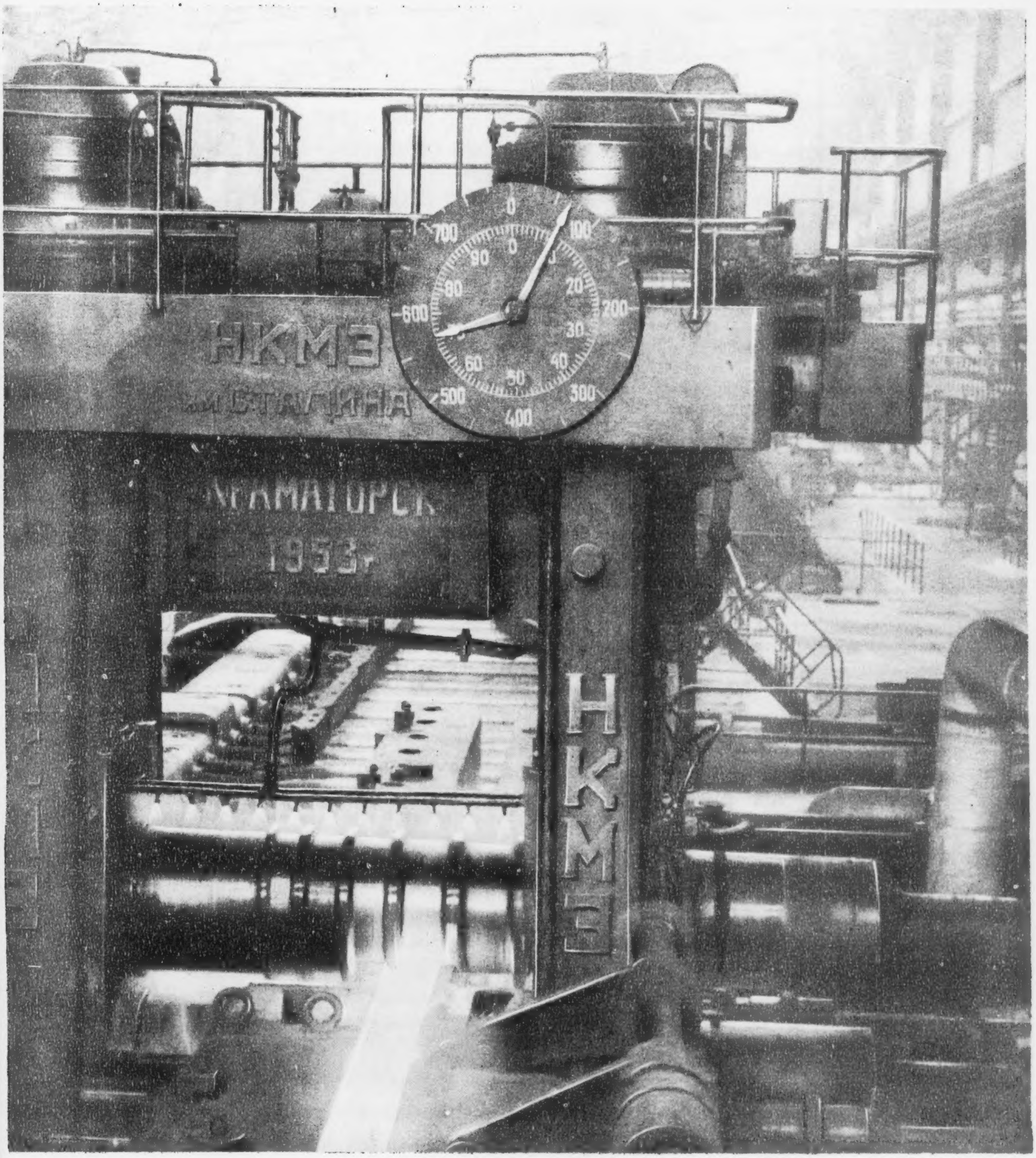


Tube billet mill at the Dzerzhinsk Metallurgical Works. Photo by I. Basilenko.

ORGANIZATION OF REPAIR SERVICES AT METALLURGICAL WORKS

The article by V. F. Ivanov, "Organization of repair services at metallurgical works," published in Metallurgist No. 5, 1957, and the comment on it by A. I. Gurvits ("Metallurgist" No. 6, 1957) have aroused considerable interest among the readers of this journal.

Publishing the comments received on the article by V. F. Ivanov, the editor would like the mechanics of metallurgical works to express their opinions on the problems raised in the article and in the discussion.

G. V. Antsyshkin

Deputy Chief Mechanic of the Zlatoust Metallurgical Works

In his article on the organization of repair services Comrade Ivanov writes that three systems of repair services organization are in use at metallurgical works: centralized, decentralized and mixed, and Comrade Gurvits, in his response to the above article, maintains that the advantages of the centralized system of equipment repair are obvious to all mechanics.

It is, however, entirely wrong to call such an organization of repairs a system.

There is only one system of organizing repair services at metallurgical, as well as at other, works in the Soviet Union – the system of planned preventive repairs (PPR) of the equipment. It constitutes a complex of organizational and technical methods of preventive character which are periodically carried out according to plan, and it is designed for maintaining the equipment in working order; the system was conceived and developed in the USSR and constitutes a part of our socialist planned economy.

What Comrades Ivanov and Gurvits call the systems of equipment repairs is nothing else but forms of organizing the system of planned preventive repairs; and Comrade Ivanov further writes in his article: "In any form of centralization the services of maintenance and duty personnel of the plant should be utilized for carrying out large scale repairs."

Indeed there are three forms of repair services organization at metallurgical works: centralized, decentralized, and mixed. The mixed form of PPR organization is a transitional one from the decentralized to a more developed, centralized form of repairs organization; with the changing conditions and the volume of production, this transitional form will be found unsuitable for ensuring the full repair of equipment, it will be gradually reorganized and in its content will approach the centralized form.

We consider it necessary to make these remarks in order to clarify the concepts of the system and the forms of organization for works repair services; otherwise, yet new "systems" of metallurgical repairs might be recommended.

Comrade Ivanov recommends three forms of centralized repairs:

1) sectional repair shops of the types existing at the Magnitogorsk Metallurgical Combine;

2) a single repair and fitting plant with specialized repair sections as at the "Azovstal" Works;

3) a single repair and mechanical plant with teams for the repairs of the main plants.

V. F. Ivanov considers that the sectional repair shops may be recommended only for very large metallurgical works which have several main plants and which posses a large quantity of machining equipment which can be partly transferred to the repair stations. Setting up of sectional repair shops at medium and large works is, in Comrade Ivanov's opinion, inadvisable, because the repair personnel would not be fully occupied during the intervals between major overhauls. For the works with a small number of main plants, Comrade Ivanov recommends the third form of centralization: a single repair and mechanical plant. It seems to us that it is impossible to agree with such suggestions because the repair sections are organized not only out of large repair and mechanical shops with a large quantity of machining equipment, part of which is transferred to the sections, but also with the machining equipment of the plant repair shops, the equipment being transferred and assembled in the sections. There is a fair number of metal cutting machines in the repair shops of the main plant (from 4-5 to 10-15). The repair sections also take over a part of the repair and duty personnel. The sections repair centers, set up on this basis, are then supplemented with new machining equipment received by the works and so become adequately large repair sections.

There is no reason to fear an inadequate occupation of repair personnel in the periods between major overhauls at large and medium works. At medium and small works many units are not operating seven days a week, each unit having its day off. The repair personnel in such cases would carry out repair and inspection jobs specified by the system of planned preventive repairs. If we add that the centralization of equipment repairs is impossible without unit-by-unit replacement of parts for which the repair personnel collects and prepares, during the days when no repairs are done, the units for the forthcoming overhaul, it then becomes obvious that the repair personnel of the sections will always be adequately occupied.

Therefore we consider that the most effective form of centralization of repair services can be adopted not only at very large works, but also at the medium ones. Moreover, a full centralization of equipment repairs is expedient and economically advantageous at small metallurgical works. In actual fact, in a decentralized organization of repair services at a small metallurgical works each main plant should have on its staff a fairly large number of repair and duty personnel to carry out the planned-preventive routine, medium and major repairs, who naturally will not be fully occupied during the plant operation. Thus it is possible to transfer a number of these workers and a part of the machining equipment to the repair and mechanical plant and to set up in it a shop for equipment repair for all the main plants. In this case, the repair of metallurgical equipment will be carried out in a fully centralized way but by the shop – a branch within the repair and mechanical plant – and not by a separate repair section. The duty and repair personnel which remains in the main plants, will carry out small jobs on the maintenance of equipment during the time between the shifts and on the check inspections according to PPR schedules.

Such an organization of metallurgical equipment repairs at small works is in our opinion the most effective, because it has the same advantages as the first form of centralization (equipment repair based on separate repair sections).

It should be borne in mind that the proposed form of repair services organization allows a much better utilization of machining equipment. The repair personnel specializes in the repair of equipment without being distracted on outside jobs (as it is done in the main plants).

Under these conditions the quality of equipment repairs will improve markedly, the length of repair time will be cut down, and unscheduled stoppages will be reduced.

In the centralized form of repair services organization the consumption of means and materials on the repair of equipment will decrease considerably. Therefore a full centralization is desirable at all metallurgical works.

G. I. Shandrenko

Sci. Res. Inst. Ferrous Metals

In the solution of the problem of the most appropriate system of repair services organization, the necessary introduction of local specialization in the production of spares and components must be taken into account.

Under the centralized system of repair services at the metallurgical works, the equipment components, exchangeable and spare parts, and units should be made in the repair and mechanical plants (casting, forging, metal construction, mechanical etc.) and handed over to a central store. Making of components and spare parts in specialized repair and mechanical plants will cost less and the parts will be of better quality than those made in numerous shops attached to production and auxiliary plants at metallurgical works.

Therefore it is impossible to accept the form of repair services centralization recommended by Comrade Ivanov where "each repair section has at its disposal a machine stock for executing simple, fairly light, and frequently exchangeable parts."

In our opinion it is expedient to have in each repair section only a very limited number of auxiliary machines (drilling, grinding, turning, etc.), necessary for fitting and assembling jobs.

The number of machines should not be the deciding factor for the adoption of an acceptable and appropriate system of repair services organization at this or that metallurgical works. It is not the number of machines but their utilization (on the basis of capacity, calendar time, actual machining time, and auxiliary work time) which is relevant, and in particular the utilization of the machines which are in mechanical shops attached to main and auxiliary plants.

In our opinion the most appropriate form of repair services centralization at metallurgical works is one which will ensure:

1) a specialized production within the works of spare parts and components of equipment in the mechanical plants of the works (in other words, the components should not be made in sectional shops or directly at the plants, but the mechanical, forge and other plants should be switched over to the large scale serial production of equipment components and spare parts);

2) a technological specialization within the works of the production of starting materials for casting, forging and thermal plants, and in steel construction plants;

3) practical specialization, within the works, of repair jobs (erecting, dismantling, fitting and assembling) on separate units, machines and plants in mechanical and assembling, repair and fitting, and other plants.

The transfer of individual specialized repair and assembling groups, shops, and plants into one large section does not contradict the idea of the centralized repair services organization, and in some cases it may be expedient.

A modern centralized system of repair services organization should provide for major overhauls according to a staggered scheme; the total of the major overhaul is then broken down into small repairs which are carried out during scheduled stoppages of the plants for current repairs.

Worn-out components and units are replaced by new ones which are delivered to the repair places from plant stores or from the central spare parts store. In this case, the repair and mechanical plants supply the storage which is also supplied with equipment components, spare parts, and units from other specialized works.

BLAST-FURNACE PRODUCTION

COMPACTING OF FINELY PULVERIZED ORE MATERIALS

G. V. Gubin

Institute of Metallurgy, Acad. Sci. USSR

At present, a considerable part of lean ores is subjected to an extensive beneficiation which results in finely ground concentrate consisting of 80% fractions below 0.06 mm. Such concentrates can be used for smelting only after compacting. The compacting, however, of the materials of less than 0.2 mm in size is very difficult.

Operating indices of sintering finely ground concentrates on sintering machines are fairly low, the main reason being a low gas permeability of the charge. Hence the metallurgists have the task of devising rational and highly efficient methods for compacting iron ores concentrates.

At present the main tendency in this field is nodulizing concentrate in drums or plate granulators with subsequent roasting of produced nodules. The method is relatively simple and shows high efficiency of the process. The quality of the nodules is much better than lumps produced by other methods of compacting.

Rotary drums and plate (disc) granulators, are the most widely used equipment for compacting of ore concentrates. The efficiency of drums and the quality of lumps depend in the main on the size of the drum, its angle of inclination, speed of rotation, amount of charge, and size of lumps.

Recently, plate granulators – round, inclined tables with cylindrical, flanged edges – began to be applied. Table granulators are lighter than drums, their speed and inclination angle is more easily controlled and the sizing of nodules according to the diameter is simple. Abroad, granulators of 1.6-5.5 m diameter are manufactured. The output of a granulator varies between 3 and 25 t/sq m depending on the type of ore. After compacting, the nodules are subjected to oxidizing roasting at 200-300°C.

Hematite crystals are formed on the surface of magnetite grains and bound the magnetite grains. At higher temperatures and with an excess of air the degree of oxidation increases, and the rate of bonding intensifies on account of the recrystallization and the growth of hematite grains.

At 1100-1200°C a slag is formed from ferrous oxide and silicon, and it wets the magnetite grains, thus slowing down their further oxidation. On cooling, the slag strengthens the ore particles.

For roasting the nodules, circular or rectangular pit furnaces are built. Such furnaces proved to be especially good for roasting of magnetite nodules. The heat in the pit furnaces is provided by hot gases from precombustion chambers. The temperature of roasting varies within wide limits; for roasting of nodules from rich concentrates the temperature is raised to 1300-1350°C, and for roasting of nodules from concentrates of low softening point the temperature of roasting should not exceed 1150°C.

A disadvantage of roasting in pit furnaces is the need for having nodules of a rather high strength because the lumps in the lower layers in the furnace are subjected to a considerable pressure. Therefore the product obtained should have a high softening temperature.

It is much better to roast the nodules on the moving sintering hearth, since the height of the nodule bed can be substantially decreased, the charging and discharging of the material simplified, and the roasting process easily controlled.

At present in the ferrous industry of some countries, sintering machines with a combined, upper and lower, sucking of air are introduced. The use of such equipment for nodule roasting will allow the elimination of the harmful effect of condensed moisture on the strength of the lumps which are at the bottom, and a considerable output increase on account of a substantial increase in the height of the nodule layer up to 1 m.

In 1954 a Stalin prize winner, Candidate of Chemical Sciences, P. I. Kanavets, proposed a catalytic chemical method for compacting finely ground metallurgical materials without the process of roasting the nodules. This method was further improved by the staff of the Institute of Metallurgy and the Institute of Fuel Minerals, Acad. Sci. USSR.

The charge for the production of nodules by the catalytic chemical method consists of finely ground iron ore and calcium oxide; coke fines, brown-coal semicoke, and plentiful coal are employed as solid reducing agents. Finely ground materials from storage bins are transferred for mixing and wetting. The charge is then fed uniformly onto the disc granulator where the catalyst solution is added in the form of a fine spray. After reaching a predetermined size, the nodules fall onto the conveyor belt and are transported to undergo the carbonization process. The nodules are treated with flue gases containing carbon dioxide from industrial furnaces, at 55-65°C in a special carbonizer. The velocity of carbonization depends on the size of the granules and on the concentration of carbon dioxide in the gases . Under ordinary conditions (without the addition of a special catalyst) the carbonization reaction of lime goes very slowly and practically ceases immediately due to the formation of $CaCO_3$ film which hinders the diffusion of CO_2. The catalyst assists in transforming lime into a soluble compound and furthers a rapid reaction between the dissolved compound produced and carbon dioxide. As a result of the reaction, the lime is transformed into $CaCO_3$ and the nodules are strengthened. The reducibility of carbonized nodules of magnetite concentrates is very high. It is explained by the presence of a solid reducer in the form of lumps and the absence of difficultly reducible fayalite. Although some deformation of nodules under load takes place at 950°C, their softening occurs at 1200-1300°C. The lumps obtained by the catalytic chemical method resist the crushing and breaking down as well as the action of moisture and the effect of low temperatures. Admittedly the strength of nodules from magnetite concentrates at KMA, when they are reduced with carbon monoxide, is somewhat decreased as a result of their volume increase. But the carbonized ore and coal nodules from concentrates of KMA and Krivoi Rog may be extensively used in electric furnaces, low-stock blast furnaces, and furnaces operated on the principle of the boiling layer.

Further research works on the preparation and testing of nodules made by the catalytic chemical process from various finely ground iron ore materials show the possibility and effectiveness of using such nodules in the modern large blast furnaces.*

The compacting of iron ore concentrates extends the possibilities of ore preparation for smelting and furthers the development of beneficiation methods. Moreover, the nodules, after strengthening, may become an excellent raw material for blast furnaces, Viberg furnaces, and other metallurgical units. The application of material prepared, in a chemical and physical respect, for the blast-furnace process is one of the important potentialities for the increase of pig iron output and fuel economy. In this sense, the production and use of compacted materials in the blast furnaces is one of the radical ways of solving the problem of ore preparation for smelting.

* As a result of carbonization of nodules the content of $CaCO_3$ increases, and heat will be required for its decomposition in the blast furnace. In this respect the catalytic chemical method of nodule production is different from the production of sintered agglometate.

IMPROVED DESIGN OF CHARGING APPARATUS

L. Ia. Matusevich

Blast-Furnace Plant of the Kuznetsk Metallurgical Combine

With the change-over to operation of blast furnaces at an increased gas pressure and the introduction of sinter into the charge, the life of the main components of the charging equipment – small and large bells, their rods and hoppers – decreased markedly; the life of the stuffing box of the distributor, the protective rings of small bell sleeve and of the receiving hopper armor fell. The employees of the KMC (Kuznetsk Metallurgical Combine), I. S. Liulenkov, I. F. Domnitskii, S. P. Kochnev, S. F. Patrin, E. Ia. Izosimov, P. E. Iakovlev, A. A. Kazantsev, A. V. Omelin, and the author produced a new design of charging apparatus in 1955.

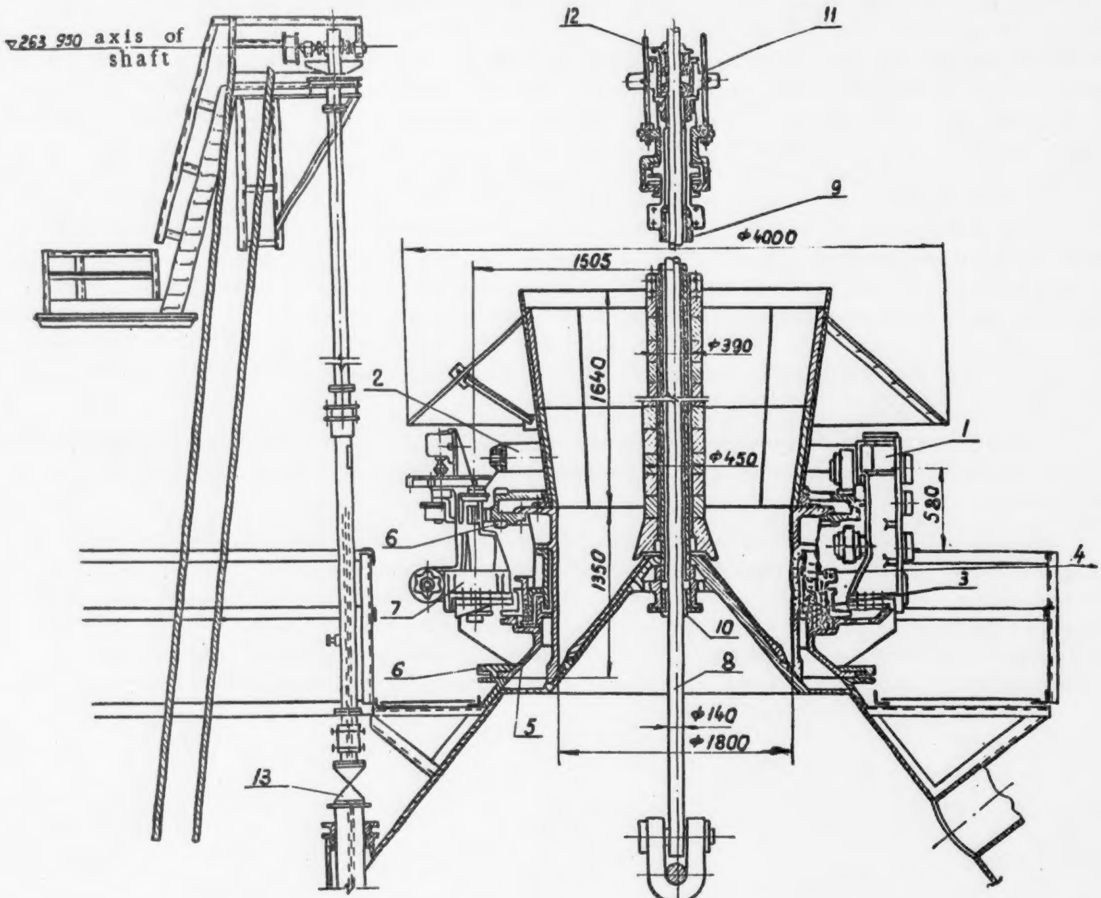


Fig. 1. Charging apparatus of KMC design:
1) roller block; 2) tightening ring; 3) stuffing box; 4) labyrinth ring; 5) base ring; 6) joining ring; 7) reduction worm gear; 8) large bell rod; 9) small bell sleeve; 10) bushing; 11) stuffing box; 12) screws; 13) plug valve.

Instead of six rollers of the charge distributor (Fig. 1) there are three blocks of rollers on roller bearings which ensure five years' operation without overhaul. The construction of the blocks makes possible their displacement in a vertical and horizontal direction during the operation of the distributor. The roller blocks are rigidly connected to each other by means of tie rods.

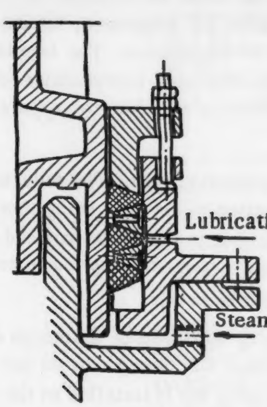


Fig. 2. Stuffing box of rotating distributor.

The stuffing box of the distributor is enclosed within three cast iron rings (Fig. 2) which are made up of six sectors with a view to facilitating assembly. For providing the passage for lubricant along the circumference of each ring, there are 12 drilling holes. Circular recesses on the inside of each ring make possible a uniform distribution of lubricant.

The labyrinth ring is made of steel 50G2. It is subjected to surface hardening and polishing in order to increase its durability. In order to prevent the ingress of abrasive dust into the stuffing box, an excessive heating, burning out of lubricant and deformation of the labyrinth ring itself, steam from the common steam main is let in at six places on the periphery and it passes through the labyrinth seal under a higher pressure than the pressure in the space between the two bells. Thus the ingress of dust into the stuffing box is prevented. Moreover, a continuous passage of steam through the labyrinth seal ensures that the stuffing box is at a constant temperature (90-100°C) at which it is not necessary to supply the lubricant continuously into the stuffing box (as in the charging apparatus of standard design), for it is adequate to replenish the lubricant once a day.

For the introduction of additional stuffing into the stuffing box there is a ring projection in the labyrinth ring. When the lower horizontal rollers of the distributor are displaced, the hopper of the small bell descends together with the labyrinth ring onto the upper part of the base ring and hermetically seals the space between the two bells. Stuffing can be added to the stuffing box during the operation of the furnace when the distributor is switched off. Such a stuffing box seal has been in operation for more than two years. During that time it was not tightened even once, and the hermetic seal remained unimpaired. There is reason to assume that this design of stuffing box will ensure unimpaired hermetic seal for 5-6 years.

The feed of steam into the space between the bells for the preventing of explosion is compulsory in all cases. In the present design of the distributor, the steam which passes through the stuffing box is appropriately used for this purpose. This steam almost immediately intersects the air stream sucked in on opening the small bell. During the operation of the equipment of this design, not a single case of gas flashing on opening the small bell was observed. The consumption of steam when it is supplied through the labyrinth seal by a special main, is reduced.

The small bell is made of high-manganese steel 13G; its contact surface is plated with sormite alloy. The treatment and working of high-manganese steel is well mastered at theCombine and does not present any technical difficulties. Owing to a small projection, 30 mm high, above the contact surface of the small bell the erosion of the surface during the descent of the charge is reduced. After one year of operation the projection of a small bell was finally worn out and only then the erosion of the contact surface began, whereas previously the erosion of the contact surface began from the very first day that the small bell was put into operation. In future such a projection will be made on the large bell also.

In order to increase the life of the small bell to three or four years, a two-layer bell is made and installed at present: its outside part is cast from chilled cast iron of 300-350 H_B hardness, and the remaining part is made of medium manganese steel 50G2. Experience showed that the armor of the receiving hopper, cast from cast iron with chilled cast iron layer of 7-8 mm, lasted for two years. There is reason to believe that the two-layer small bell with chilled cast iron outside layer of 30-35 mm will serve for at least three years. The plating of all the surface of the bells with hard alloys is not possible at the majority of works because of the lack of necessary equipment and the costliness of that process. The two-layer casting is considerably cheaper and can be adopted at any works.

The shell of the gas shut-off is made detachable in order to speed up the alignment of the small bell during the assembly when the large bell is out of alignment. The flange of the base ring is joined by a ring to the stationary part of the gas shut-off shell. After the large bell has been centered, the bracing bolts of the ring are

loosened, the small bell is lowered a few times and the distributor is set rotating while the bells are shut. Thus, the hopper of the small bell aligns itself with respect to the large cone rod. After the alignment, the bracing bolts of the ring are tightened again. The centering of the distributor takes 20-30 minutes. The annular gap of 60 mm width between the flange of the base ring of the distributor and the recess of the ring compensates for any inaccuracies during the preliminary test assembly. Hence, a general test assembly of the charging apparatus is superfluous.

The distributor in the charging apparatus is rotated by a vertical worm reduction gear. The reduction gear is connected through a fastening joint to the tie rods of the roller blocks, vibration of the gear being thus prevented. The seal between the large bell rod and the small bell sleeve is provided by bushes in the lower part and by stuffing boxes in the upper part. The stuffing box ring is tightened by means of screws and a spring in order to ensure a continuous hermetic seal between the stuffing box and the large bell rod.

For the prevention of gas leaks through the indicator holes, in the charging apparatus of the design described, there are closed type indicators which exclude any escape of gas from the furnace and ensure normal gas conditions in the furnace top. For the exchange of the rod or the inspection of its end, a plug tap is installed in the lower part of the tube and it serves for the establishment of normal conditions for the repair workers. The level of the tap plug serves as a base for the adjustment of the indicators' position.

The adoption of the charging equipment of the new design on all blast furnaces of the KMC made possible a saving of more than 400,000 rubles on account of better durability, reduction in the maintenance work and the elimination of gas escape through the indicators' apertures. Furthermore, working conditions of the maintenance and operational personnel of the furnace top improved markedly.

In 1958, during the completion of the current overhaul, all furnaces will be equipped with two-layer small bells and the hardness of the armor of the receiving hopper made of a special chilled cast iron will be raised to 600 H_B, and thus it will be possible to change to two-year intervals between current overhauls.

PRESSURIZING OF SCALE-CAR CABINS

A. A. Krivosheev and A. G. Geiko

Dzerzhinsk Works

In spite of the complete mechanization of uptake and weighing of charge materials in the blast-furnace plant of the Dzerhinsk Works, working conditions of scale-car operators were until recently unsatisfactory as the air under the bunkers was full of dust. The change-over of the blast furnace to operation with hot sinter (400-450°C) worsened the working conditions still more.

In 1955-1957, nine pressurized scale-car cabins with cleaned air intake were installed at the Dzerzhinsk Works.

The pressurized cabin is of welded construction from 3 mm iron plate.

Special observation windows are made in the side walls of the cabin (two on each side) for the operators to watch all the operations on uptake and charging of materials into the skips.

The movement of the scale-car down the line of the bunkers can be watched through the glass door of the cabin. The observation windows are made of ordinary glass.

In the lower part of the cabin a wooden platform is set up. The control equipment of the scale-car is inside the cabin (Fig. 1). It allows the operator to carry out all the charging operations without leaving the cabin.

For cleaning the intake air to the cabin a special dust-removing tank is mounted on the base frame of the car scale; the tank is connected through a 300-mm diameter tube with the fan apparatus which consists of the fan, a motor and a duct through which air is supplied into the cabin. The dust-removing tank (Fig. 2) is made of 2-mm thick iron plate. The dimensions of the tank are: 1350 × 1000 × 500 mm. Inside the tank, approximately

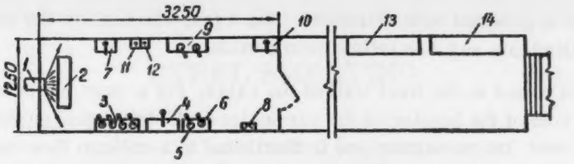


Fig. 1. Layout of equipment in the scale-car cabin:
1) air intake into the cabin from the fan equipment; 2) scale dial; 3) control levers of pneumatic cylinders for lifting of reduction gears; 4) control of scale-car movement; 5) signal whistle; 6) brake lever; 7) control of rotational mechanism of the drum shut-off of the bunkers; 8) manometer, indicating the pressure of compressed air in the air duct; 9) control levers of pneumatic cylinders for opening the bunker doors; 10) pneumatic switch-off relay of compressor; 11) automatic switch-on of the fan for air supply to the cabin; 12) automatic switch-on of the pump for filter washing; 13 and 14) switch panels.

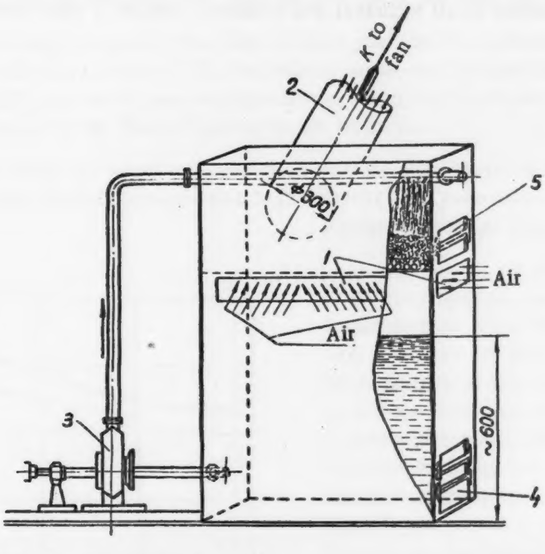


Fig. 2. Dust-removing tank:
1) air intake windows; 2) duct for air supply to the cabin; 3) pump; 4) hatch for cleaning the dust from the tank; 5) door for coke charging.

at half its height, there is a metal grid made of 10-mm diameter rods, with 15-mm spaces between the rods. A layer of coke (20-40-mm fraction) 120-150 mm high is on the grid. The lower part of the tank is filled with water, up to a level 150-200 mm below the grid.

Special windows for air intake are cut in the tank walls on both sides, between the grid and the water level. The air on passing through the moist coke layer is cleaned of dust and is delivered to the cabin by means of the fan. The water for washing the coke filter bed is delivered from the lower part of the tank by means of a pump 700-40 through a 1.5 in diameter tube which passes over the filter bed along its whole length. Holes of 3 mm diameter are drilled along the underside of the tube over approximately one third of its circumference. In the lower part of the dust remover there is a hatch for cleaning the collected dust from the tank, and in the upper part a hatch for charging ground coke into the tank.

Ventilation equipment is provided with "Sirokko" No. 4 fans which allow the establishment of such an air pressure in the cabin as to eliminate any dust inflow from outside.

The air from the fan is passed to the front wall of the cabin. For a more uniform distribution the air stream is directed against the back side of the housing of the car-scales dial. Impinging on the walls of the dial housing the air stream is dispersed, loses its momentum and is distributed in a uniform flow over the whole cabin. Thus the operator always feels a light refreshing breeze.

The purity of the air pumped into the cabin depends on the maintenance of the dust remover. A thorough maintenance, timely washing of the filter bed and an uninterrupted fan operation are the necessary conditions for obtaining adequately pure air. The temperature of the air delivered to the cabin fluctuates usually within the limits of 18-20°C.

The filter grid and the tank are cleaned once a week to remove adhering dust. The filter bed is washed with water every 5-6 chargings. The coke on the grid is changed on major and intermediate overhauls of the scale-car.

In order to facilitate the maintenance and repairs, all the fan equipment is situated on the main frame of the scale-car along with the air compression machinery.

The setting up of a dustproof cabin on the working platform of the scale-car made possible a considerable improvement in working conditions of the operator, an increase in his operating efficiency and, in the end, the attainment of a further improvement in all technical and economic indices of blast-furnace operation.

STEEL SMELTING

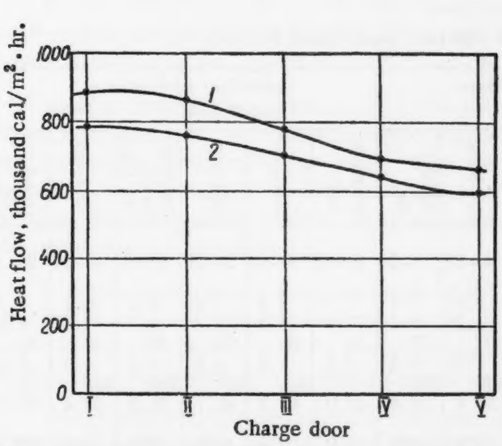
USE OF LOW-PRESSURE, HIGH CALORIFIC VALUE HOT GAS IN OPEN-HEARTH FURNACE OPERATION

Prof. Doctor of Tech. Sciences V. S. Kocho, V. I. Grankovskii, Iu. D. Molchanov and E. A. Ploshchenko

At present, the gas-fired open-hearth furnaces are, as a rule, fired with a mixture of coke-oven and blast-furnace gases, the flame being carburetted with fuel oil or a mixture of three gases – coke-oven, producer and blast-furnace. The low calorific blast-furnace gas is used to impart rigidity and flatness to the flame. The blast-furnace gas, however, lowers the theoretical temperature of combustion because the heat content of 1 standard cu. m of the combustion product of the blast-furnace gas is less than that of the coke-oven gas.

The elimination of a large amount of the blast-furnace gas, which constitutes approximately half the volume of the mixed gas, would allow an increase of the theoretical combustion temperature, a reduction in the volume of combustion products, and, as a result, the establishment of substantial draft potentialities, i.e., the potentialities for further intensification of the thermal process in the furnaces.

The Voroshilov Metallurgical Works in conjunction with the Kiev Polytechnical Institute carried out research work on the operation of open-hearth furnaces with high calorific gas (coke-oven gas alone), turbine air being supplied to the gas port.



Variation in direct heat flow along the bath of the 500 ton open-hearth furnace during the finishing period (thermal regime: coke-oven gas 6000 cu m/hr, fuel oil 300 kg/hr, turbine air 5000 cu m/hr):
1) blast-furnace gas consumption equal to 5000 cu m/hr; 2) same, equal to 7000 cu m/hr.

The result, on the completion of the first stage of the investigation, was the introduction, on all furnaces of the open-hearth furnace plant, of operation with a reduced (by half) amount of blast-furnace gas: 3000 cu m/hr in 250-ton furnaces and 4500 cu m/hr in 500-ton furnaces, the amount of coke-oven gas remaining almost the same as previously (the amount of coke-oven gas supplied to the plant remained unchanged). Specific consumption of fuel fell by a quantity approximately equivalent to the amount by which the blast-furnace gas was reduced.

Moreover, on account of increased outlet cross section of the gas port (from 0.44 to 0.48 sq m at 250-ton furnaces) and the reduction of blast-furnace gas consumption, the same high temperature of heating the gas- and the air checkers was attained (about 1350°C).

The application of turbine air made it possible to operate at an air excess coefficient equal to unity during all periods of steelmaking, apart from the melting period and the period of violent gas evolution during the ore boil.

On account of the volume reduction of combustion products due to the reduced consumption of blast-furnace gas and air, the dust carry-out decreased considerably. Whereas previously the stationary slag pockets were filled up in 200-250 heats, now a complete filling-up of removable slag pockets (smaller in volume) takes place after about 400 heats.

TABLE 1

Mean Data on the Thermal Operation of a 500-ton Open-Hearth Furnace

Item	Operation without blast-furnace gas						Operation with blast-furnace gas					
	preparation	charging	heating	pouring-in of pig iron	melting	finishing	preparation	charging	heating	pouring-in of pig iron	melting	finishing
Duration, hr—min.	0—20	2—05	1—45	0—33	4—47	2—45	0—15	2—02	1—57	0—36	4—50	2—20
Consumption, thousand standard cu m/hr												
coke-oven gas	5.2	8.8	8.6	7.07	5.63	5.93	4.7	7.88	7.38	5.88	4.93	5.48
blast-furnace gas	—	—	—	—	—	—	7.25	7.88	8.00	7.63	7.63	8.00
fan air	26	41.7	34.3	33.0	32.7	32.7	20,0	38.0	31.5	29.33	30.75	28.75
turbine air	3.55	4.7	4.77	4,73	4.78	4.72	3.1	4.35	4.5	4.36	4.50	4.20
Consumption of fuel oil, kg/hr	300	350	250	400	470	470	240	387	387	460	485	435
Thermal load, million cal/hr	22.7	37.4	36.7	32.2	28.3	29.4	28.7	44.8	42.8	37.0	37.6	37.0
Temperature, °C												
roof	1590	1560	1620	1640	1645	1680	1580	1540	1600	1570	1590	1670
top of air checkers	1285	1260	1247	1275	1330	1340	1280	1260	1270	1300	1320	1350
top of gas checkers (max)												
right	1285	1285	1275	1295	1360	1310	1127	1087	1087	1082	1087	1100
left	1330	1320	1290	1320	1350	1390	1127	1120	1127	1120	1140	1140
flue gases	497	520	530	520	570	610	355	380	360	415	400	360
Coefficient of air excess,	1.14	1.10	0.93	1.01	1.17	1.1	0.73	0.86	0.73	0.77	0.82	0.77
Weight of heat, t	503.7						489.8					
Specific consumption of conventional fuel, kg/t	108.7						135.6					

TABLE 2

Average Data (from 10 heats) of the Thermal Operation of the 500-ton Open-Hearth Furnace

Item	Operation without blast-furnace gas						Operation with blast-furnace gas					
	preparation	charging	heating	pouring-in of pig iron	melting	finishing	preparation	charging	heating	pouring-in of pig iron	melting	finishing
Duration, hr - min	0—18	2—55	2—23	1—05	4—20	2—26	0—18	3—05	2—00	0—42	4—35	2—45
Consumption, thousand standard cu. m/hr												
coke-oven gas	5.1	7.3	7.1	5.8	5.8	5.6	5.8	8.0	7.8	6.2	6.1	5.9
blast-furnace gas	1.4	0.3	0.6	0.7	1.3	1.6	4.2	4.3	4.3	4.3	4.3	4.2
fan air	25	35	31	30	32	29	29	36	36	32	33	32
turbine air	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5
Consumption of fuel oil, kg/hr	30	500	360	460	440	360	—	560	380	380	460	370
Thermal load, million cal/hr	23.9	36.6	34.3	27.4	30.4	28.8	29.0	42.3	40.7	32.9	34.4	32.3
Temperature, °C												
top of air checkers	1190	1190	1200	1230	1270	1220	1200	1250	1260	1290	1300	1280
top of gas checkers (max)												
right	1320	1310	1310	1310	1300	1310	1300	1300	1300	1300	1320	1300
left	1320	1310	1320	1310	1300	1320	1300	1320	1320	1310	1290	1290
Weight of heat, t	506						501					
Specific consumption of conventional fuel, kg/t	120.1						141.9					

The reduction in the consumption of blast-furnace gas increased the temperature in the working volume of the furnace by 20-40°C. Direct heat flow increased substantially. The results of measurements made with the thermal probe of VNIIT design on the 500-ton open-hearth furnace during finishing period at a constant

consumption of coke-oven gas (6000 cu m/hr), fuel oil (300 kg/hr), turbine air (5000 cu m/hr) and alternative consumption of blast-furnace gas (5000 cu m/hr and 7000 cu m/hr) are given on the diagram. It is seen from the diagram that the direct heat flow along the axis of the furnace increased on the average by 50-70 thousand cal/sq m•hr.

In order to test the possibility of operation on heated-up pure coke-oven gas, three test heats were carried out in a 500-ton open-hearth furnace.* The averaged data from the three experimental heats are given in Table 1. For comparison, the averaged data from four heats carried out with blast-furnace gas are also given in the table.

It is seen from the table that the duration of the heat remained practically the same and the consumption of conventional fuel per 1 ton of steel decreased by 27 kg.

At the end of the third heat the luminosity of the flame fell markedly, the heating of metal being considerably impaired.

It is known that the luminosity of the flame is imparted to it by carbon black formed on the decomposition of methane, heavy hydrocarbons and tars. Intensive decomposition of methane, up to 80%, takes place in the range of temperatures 600-800°C and is completed at 1200°C. Approximately in the same range of temperatures the reaction $C + CO_2 = 2CO$ takes place in the direction of CO formation. Assuming that the gas, as a rule, is at 1000°C, the fall in luminosity of the flame (on account of combustion during turbine air supply or on account of air being sucked in along the gas stream) should take place also at the usual temperature of gas regenerators. But such a phenomenon is not observed.

It can be suggested that at high temperatures the metallic oxides which are on the surface of the checker brick react with carbon black and remove it from the gas. Another suggestion also is most probable; on prolonged overheating the lower brick rows of the gas regenerator are at about 800-900°C and at that temperature an almost complete decomposition of methane takes place and heavy hydrocarbons decompose even at lower temperatures. Because of a small volume of gas and a relatively large volume of gas checkers the heating of gas to those temperatures takes place even in the lower rows of the checker and there the carbon black is deposited on the surface of bricks and is then taken up with the flue gas.

It was established in the experimental heats that there is an optimum temperature above which it is undesirable to heat the gas, since it loses its luminosity. On maintaining this temperature of the gas regenerator without blast-furnace gas, a good heating of metal is attained and the specific fuel consumption is reduced.

Because of the damage to the lining of the gas ports of the 500-ton furnace it became possible to operate on pure coke-oven gas. The combustion products leaving through the gas checkers were undergoing, in this case, a preliminary cooling in the port. Thus, the overheating of the gas checkers and the loss of flame luminosity were eliminated. The furnace was operated under these conditions for about a month and a half. Operating data are given in Table 2. For comparison, data for 10 heats carried out with the usual blast-furnace gas consumption are also given.

It is seen from Table 2 that the duration of the heat on pure coke-oven gas was reduced on the average by 0.7 hour and the consumption of conventional fuel decreased by 21.8 kg/ton of steel. It indicates that the change-over to operation with high calorific gas makes possible a substantial improvement in the thermal operation of the furnace, a decrease in fuel consumption and a reduction in heat duration.

The change-over to operation with pure coke-oven gas is advantageous and timely for all the open-hearth furnaces which operate either with or without oxygen, because the change makes possible a substantial increase in furnace output and a reduction in specific fuel consumption. In the case of furnaces with small volume of checkers and where there is no potential reserve for draft increase, the change to operation with pure coke-oven gas allows an improvement of the thermal operation of the furnace.

For the industrial application of the above method of operation on mixed gas it is necessary to reduce the cross-sectional area of the gas port in order to reduce the consumption of the combustion products in the gas flow by 20-30% compared with the present one and thus to prevent the overheating of gas checkers. Later on a general reduction in the volume of the gas checkers may be possible.

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* With the object of reducing coke-oven gas losses in the flue during the valve change, blast-furnace gas is automatically let into the working space of this furnace. The supply of blast-furnace gas is cut off immediately after the valve change.

It should be stressed that operation without blast-furnace gas is possible only in furnaces which operate with high speed compressed air being delivered to the gas port. Blast-furnace gas should be supplied in order to reduce the disruption of the flame in the furnace during the valve change; it is easily done on the furnaces with separate delivery lines for coke-oven and blast-furnace gases to the furnace valves.

When gas of high calorific value is employed, the necessity of equipment for gas pressurizing is eliminated and there is no need for increasing the fuel oil consumption.

EFFECT OF CASTING EQUIPMENT REFRACTORIES ON THE CONTAMINATION OF BALL-BEARING STEEL

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Nonmetallic inclusions in ball-bearing steel are of very diverse origin and consist usually of oxides, spinels, silicates, glass-like inclusions, and sulfides.

An investigation was carried out at the "Electrostal" Works in collaboration with the Moscow Steel Institute and and All-Union Scientific Research Institute of Refractories, with the object of selecting refractories— for linings of ladles, spouts, and stopper rods — which are the most durable and cause the minimum contamination of steel.

The effect of the refractories used for ladle and spout linings. Chamotte, kaolin and high-alumina refractories were used for the lining of ladles and spouts. Their characteristics are given in Table 1.

On preparation of refractories, an isotope of calcium (Ca^{45}), in the amount of 150 mC per one ton of refractory material, was introduced into the refractory mass.

The ball-bearing steel was made in 20-ton electric arc furnaces, according to the method accepted at this works. The steel was bottom-poured into 500-kg ingots which were rolled into 70-90-mm sections. During rolling, samples were cut from the upper, middle, and lower part of the ingots from the first, third, and fifth set of molds, for testing for contamination with inclusions. The amount of nonmetallic inclusions was determined on a large number of samples according to the GOST 801-47 scale as well as by electrolytic separation of inclusions. The amount of radioactive calcium isotope was evaluated in the separated inclusions.

The results of the determination of metal contamination with inclusions, as well as the erosion of the refractories, are given in Table 2; it is seen from the table that the steel cast from the ladle with chamotte lining was of the highest contamination. Steel poured from a high-alumina ladle has the least amount of inclusions. The contamination of steel, poured from the kaolin ladle, is almost the same as that of steel from high-alumina ladle, but steel poured from the kaolin ladle is cooler than steel from other heats (by 30°C on the average). With the decrease in the temperature of the metal the contamination decreases substantially.

The amount of contaminations showing radioactivity is largest for the chamotte lining and smallest for the high-alumina lining.

The data on the erosion of refractories during the hot hour also indicate a smaller erosion of high-alumina and a considerably higher erosion of chamotte lining.

Thus, out of the tested linings of ladles the best results, with respect to steel contamination with inclusions, were obtained for high-alumina lining. On testing of spouts with various linings, practically no difference was found in the inclusions content.

* Engineers V. S. Nikolskii, V. S. Laktionov and a representative of the Gisogneupora, S. D. Skorokhid, took part in this work.

The effect of refractories used for fountain assembly. Chammotte, high-alumina, kaolin and graphite-chammotte refractories were employed for the fountain assembly (funnel, fountain lining, and nozzle); the characteristics of the refractories are given in Table 3.

TABLE 1

Characteristics of the Refractories in Ladles and Spouts

Refractory	Composition of material, %		Chemical composition, %		Refractoriness, °C	Apparent porosity, %	Compression resistance, kg/sq cm	Temperature at the beginning of deformation under the load of 2 kg/sq cm, °C
	Chamotte	Clay	$Al_2O_3 + TiO_2$	Fe_2O_3				
Chamotte	75	25	35.0	1.5	1670	19.2	250	1370
Kaolin	85	15	44.4	1.1	1770	10.5	700	1415
High-alumina	85	15	75.0	0.5	1850	5.6	1000	1550

TABLE 2

Results of Tests on Various Refractories for Ladles and Spouts

Refractory	Number		Mean temperature of metal on tapping, °C	Metallographic testing						Amount of inclusions, %		Oxygen content, %	Erosion of refractory during the hot hour, %	
				Oxides		Sulfides		Globules						
	Heats	Samples		Mean mark	Amount of samples with mark higher than 2.5, %	Mean mark	Amount of samples with mark higher than 2.5, %	Mean mark	Amount of samples with mark higher than 2.5, %	Radioactive	Separated		Mean	Maximum
Ladle Tests														
Experimental chamotte	8	288	1560	1.81	14.6	1.45	2.7	0.32	5.9	4.6	0.010	0.0032	6.0	11.0
Kaolin	10	360	1519	1.62	8.1	1.48	1.8	0.17	3.3	2.7	0.011	0.0026	3.6	4.4
High-alumina	10	360	1550	1.58	8.9	1.45	3.1	0.16	1.7	1.5	0.010	0.0025	2.8	3.6
Ordinary chamotte	16	576	1554	1.78	12.2	1.53	0.5	0.25	3.3	—	0.010	0.0027	3.4*	10.2*
Spout Tests														
Chamotte	5	180	1550	1.68	11.5	1.42	2.2	0.25	2.2	0.4	0.010	0.0030	—	—
Kaolin	5	180	1546	1.77	10.6	1.59	3.4	0.10	1.7	0.7	0.010	0.0026	—	—
High-alumina	6	216	1546	1.89	14.4	1.57	2.8	0.40	6.0	0	0.009	0.0024	—	—

* Mean erosion over a year.

The tests were carried out on five heats; for each of them five mold sets were lined with different refractories and were re-used in turn in the sequence of pouring (the placing of experimental fountains is indicated in Table 4). Mean temperature of the metal from five heats was 1583°C.

Samples were taken from the upper, middle and lower part of two ingots (on rolling to rounds of 70-90 mm) from each mold set and they were analyzed for inclusions content (in the same way as the metal poured from ladles lined with various refractories). The effect of each refractory was tested on 90 samples (18 samples from one mold set of a given heat).

Results of the investigations (Table 5) show that the fountain refractories under the experimental conditions have a smaller effect on the contamination with inclusions than the ladle refractories.

TABLE 3

Characteristics of Fountain Refractories

Refractory	Composition of material, %: Chamotte	Clay	Kaolin	Graphite	Chemical composition, %: SiO_2	$Al_2O_3+TiO_2$	Fe_2O_3	CaO+MgO	Carbon	Refractoriness, °C	Apparent porosity, %
Experimental chamotte	50	30	20	—	57	38	1.3	1.4	—	1735	18.6
Kaolin	50	25	25	—	56	40	1.3	1.2	—	1750	24.0
High-alumina	50	30	20	—	44	52	1.3	1.4	—	1780	19.5
Graphite-chamotte	35	40	—	25	45	29	1.9	1.6	20	1900**	31.3
Ordinary chamotte	50	25*	—	—	—	Calculated 52.0	—	—	—	1780	24.5

* *Technical alumina.*

** In reducing medium.

TABLE 4

Sequence of Filling the Test Fountains

Heat	Position of the bottom plate with the set of molds in the order of pouring				
1	Experimental chamotte	Kaolin	Graphite-chamotte	High-alumina	Ordinary chamotte
2	Kaolin	Graphite-chamotte	High-alumina	Ordinary chamotte	Experimental chamotte
3	Graphite-chamotte	High-alumina	Ordinary chamotte	Experimental chamotte	Kaolin
4	High-alumina	Ordinary chamotte	Experimental chamotte	Kaolin	Graphite-chamotte
5	Ordinary chamotte	Experimental chamotte	Kaolin	Graphite-chamotte	High alumina

TABLE 5

Results of Tests on Fountains Lined with Experimental Refractories

Refractory	Number of samples	Metallographic testing: Oxides: Meak mark	Oxides: Amount of samples with higher than 2,5, %	Sulfides: Mean mark	Sulfides: Amount of samples with mark higher than 2.5,%	Globules: Mean mark	Globules: Amount of samples with mark higher than 2.5,%	Amount of inclusions, %: Radio-active	Separated	Oxygen content,%
Experimental chamotte	90	1.47	3.4	1.61	1.1	0.92	3.3	0.13	0.0087	0.0025
Kaolin	90	1.41	4.4	1.70	12.2	0.84	7.8	0.13	0.0091	0.0024
Graphite chamotte	90	1.44	4.1	1.65	3.3	0.67	1.1	0.13	0.0093	0.0023
High-alumina	90	1.42	5.6	1.58	7.8	0.57	0	0.13	0.0083	0.0024
Ordinary chamotte	90	1.55	7.8	1.73	13.3	0.93	6.7	—	0.0096	0.0027

In the steel cast in the fountain lined with ordinary chamotte refractory more oxide and globular inclusions were found than in the steel case in other refractories. In these tests (similarly to the tests on ladles with various lining) it was found that the percentage of radioactive inclusion is closely connected with the mean mark of oxides which characterizes the content of large inclusions on the basis of metallographic analysis.

The following conclusions can be drawn on the basis of the above investigations:

1. Out of the investigated three types of refractories (chamotte, kaolin and high-alumina) for the lining of ladles and spouts, the least contamination of metal with inclusions and the highest durability was obtained on using high-alumina refractory which contained 72-75% Al_2O_3 and was of 5.6% porosity (mean oxide mark in heats: from high-alumina ladle – 1.58; kaolin – 162;* chamotte – 1.82).

2. The amount of radioactive inclusions from the ladle lining varies in various samples within the limits of 1.1 to 7.8% and depends on the lining of the ladle and the temperature of the metal. The higher the temperature, the larger amount of radioactive inclusions in steel, i.e., with the increasing of temperature the contamination of steel increases on account of the erosion of the ladle lining.

3. The investigated refractories (chamotte, graphite-chamotte, kaolin, high aluminous) in the stopper rod have little effect on the content of nonmetallic inclusions in steel.

On the basis of metallographic analysis, the heats poured into the molds made of ordinary chamotte brick from the Borovichskii Combine have the greatest comtamination.

AUTOMATION OF THE OPEN-HEARTH FURNACE WITH USE OF ELECTRONIC EQUIPMENT

P. G. Baranovskii and Ia. S. Pinus

KIP and Automation Plant of the Kuznetsk Metallurgical Combine

Shortcomings of some schemes of automatic control of the heating regime in the open-hearth furnace lie in the lack of intercoupling between separate controls. The change of assignment in the course of a heat for each control in these schemes has to be done manually and the correctness of the assignment setting depends on the qualification of the operator.

The open-hearth furnaces of the Kuznetsk Metallurgical Combine were previously equipped with the system of automatic intercoupled program control, proposed by Kashtial. As a result of design defects and because of inconveniences and difficulties in changing the type of intercoupling of the controls, this system was modified several times at the KMC and at other works. In recent years the system of intercoupled programmed control was not used (except for a few furnaces at the KMC) because no reliable intercoupling of the controls was found.

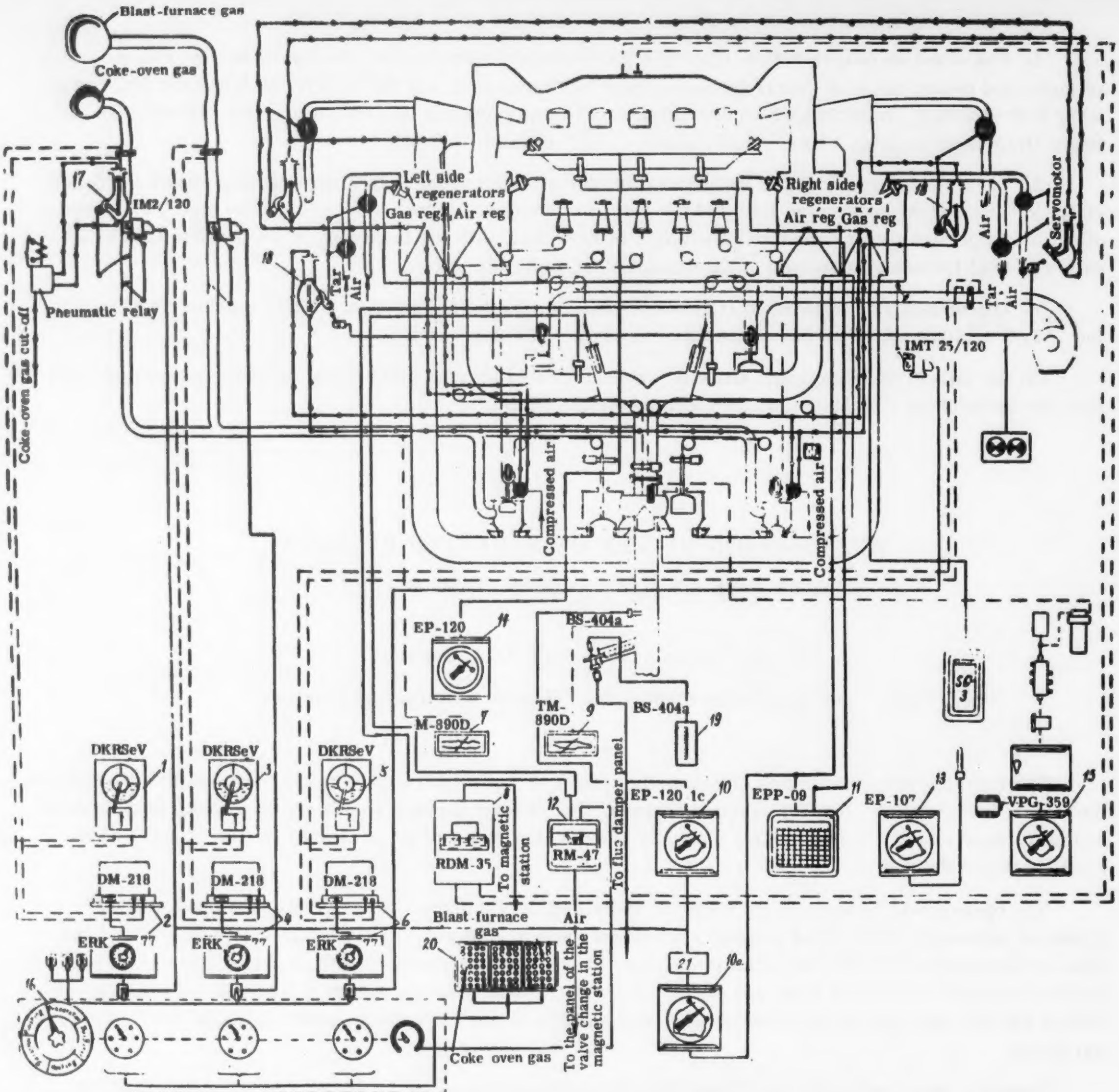
Extensive use of measuring and controlling electronic equipment in the metallurgical industry made it possible to make the system of intercoupled programmed control reliable and simple.

The application of electronic controls in the open-hearth furnace plants of the KMC began in 1955. The first attempts to use standard ERK-77 electronic quantity controls in conjunction with DM-218 differential manometers, for the control of the input of gases, gave no positive results. The control of the uniformity of gas input at the ordinary fluctuations of pressure in the mains was unsatisfactory.

Good results were obtained on testing a modified ERK-77 scheme without the dynamic compensation unit.

The system of intercoupled programmed control of the heating regime with the application of electronic quantitative controls without the dynamic compensation units was installed at the end of 1956 on one of the open-hearth furnaces fired with mixed coke-oven and blast-furnace gases. The system so far intercouples only the controls of gas and air input and valve change with some parameters which determine the thermal conditions in the working space of the furnace and the regenerators.

* The temperature of the metal poured from the kaolin ladle was, on the average, 30°C lower than that of the metal poured from the ladles with other linings.



Basic scheme of thermal control and automatic adjustment of the thermal regime in the open-hearth furnace: 1) metering of coke-oven gas consumed; 2) control of coke-oven gas-input; 3) metering of blast-furnace gas consumed; 4) control of blast-furnace gas input; 5) metering of air input; 6) control of air input; 7) measurement of mixed gas pressure; 8) measurement and control of pressure in the furnace; 9) measurement of pressure drop in the flue; 10) measurement of the flue gas temperature; 10a) control of the rate of heating of the air regenerator top; 11) measurement of the temperature of the top in the gas and the air regenerators; 12) control of the time interval between the valve change; 13) measurement of the temperature of liquid steel; 14) measurement of roof temperature; 15) analysis of combustion product (O_2); 16) selector of thermal loads; 17) coke-oven gas cut-off; 18) change-over switch of air, entering the ports, and of tar for carburetting; 19) position of the flue damper; 20) programming apparatus (commutator); 21) memory apparatus.

The system of thermal regime control consists of three quantitative electronic controls operated by a common command control (thermal load selector); programming device, determining the assignment of each control; three differential manometers, type DM-218; electrical equipment which operates the throttle valves: for gas – IM-2/120, for air – IMT-25/120; and the control of the time interval between the valve change (see diagram).

The load selector is a special switch with which the operator sets the initial assignment task for the fuel and the valve-change controls. These assignments are automatically adjusted in the course of furnace operation depending on the thermal conditions and the form of gas combustion in the furnace.

The programming apparatus is a Swedish type commutator on which the most effective ratios of coke-oven gas, blast-furnace gas and air for a given thermal load are set up in advance (the program provides for the change in the amount of blast-furnace gas side by side with the change in the amount of coke-oven gas).

In the course of the heat, the operator determines the total thermal load of the furnace (without taking tar into account) for a given period of the heat and sets the selector of thermal load in the corresponding position. According to the assignment, the controls ensure the necessary input of coke-oven gas, blast-furnace gas, and air. Simultaneously with the change in thermal load, the optimum time interval between the valve change is set up automatically.

In the new system the operator sets only the maximum value of the thermal load necessary for a given period; the remaining operations on the determination of the required quantities of gases which constitute the mixture, the air for combustion, as well as the time interval between the valve changes, are carried out automatically.

As it is not possible to determine in advance the losses of gas and air entering the furnace and the amount of combustible gases (CO) liberated from the bath during melting and refining, for the appropriate combustion of fuel an apparatus is necessary which would control the combustion process and adjust the completeness and form of combustion of gases in the furnace in order to attain a possible maximum temperature and the best possible luminosity of the flame.

The automatic adjustment of the input of gas and air and of the thermal load is generally brought about by two factors: the temperature of the products of combustion and the speed and extent of heating-up of the top of the regenerators.

The air input is adjusted according to the temperature of the combustion products leaving the regenerators and the speed and absolute value of the temperature of the heating-up of the top of the regenerators.

If the fuel is not completely burnt in the furnace and the combustion continues over the regenerators and in the flue, then the flue gas temperature increases and so does the temperature of the regenerators (owing to brightening effect). On the combination of such factors the air input to the furnace is increased automatically by 2000-4000 cu m/hour. When the furnace has excessive thermal loads according to the process conditions, the heat in the furnace is not fully utilized. Then the regenerators become quickly overheated. The electronic control of the heating-up speed of the air regenerators, by integrating the area under the temperature curve, automatically cuts down the fuel input to the furnace if the given area, reached in a time interval, is smaller than the predetermined one.

In cases when the furnace is not working symmetrically (the regenerators on one side at an elevated temperature), the control limits the amount of fuel input to the furnace during the operation on the hot side thus causing an increase in fuel supply during the operation on the cooler side. The system memorizes on which side the fuel input is corrected and adjusts the fuel input immediately after the valve change. The heating-up of the regenerators equalizes very quickly if the discrepancy was not caused by fuel losses on the way to the combustion space. If the fuel losses on both sides are different the system provides for maintaining a different input of fuel on the right and left sides until the losses are eliminated.

The operation of the system since December, 1956, indicates that although it requires further improvement, it is reliable, it adequately ensures accurate control of fuel input to the furnace in accordance with a predetermined program, it allows a simple adjustment (if it is necessary depending on the deterioration of the furnace) of the ratio of gases and air, and it is much more convenient to operate than the system with hydraulic controls (cleanliness, absence of the noise of oil pumping equipment, etc).

Furthermore this system can be easily included in the scheme of complex control, and self-adjusting control systems can be established on its basis, thus ensuring an efficient operation of the open-hearth furnace.

Good thermal and technical indices were attained during the operation of the furnace with the electronic equipment. Thus the mean duration of the heat over 5 months of 1957 was 8 hours 57 minutes, and fuel consumption was 166.6 kg/t of steel. The corresponding figures for 1956 were 9 hours 07 minutes and 167.5 kg/t.

Thus, the adoption of electronic equipment in the system of programmed control of the thermal regime of the open-hearth furnace fired with a mixture of coke-oven and blast-furnace gases, resulted in an improvement in furnace operation and opened possibilities of a further improvement in furnace control by developing self-adjusting automatic control systems.

MOLDS OF A NEW DESIGN

A. S. Korzhavin

The Dzerzhinsk Works

Molds of a new design, differing from the ordinary ones in the degree of taper of the sides, the shape of the bottom and the profile of the corners (Fig. 1), were tested at our works.

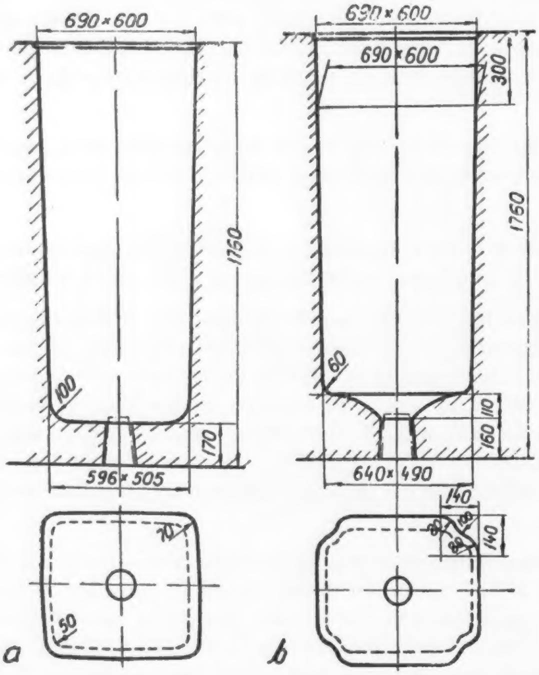


Fig. 1. Molds:
a) ordinary design; b) new design.

The upper part of the new mold (Fig. 1,b) is cast with parallel walls over a length of 300 mm. The taper of the sides on the major axis constitutes 2.1% and on the minor axis – 4.6%.

The profile of the corner between the walls constitutes a combination of three arcs (Fig. 2). The profiles of the corners of the ordinary molds are shown by the dotted line. The bottom is made in the form of a "wine-glass." The outside view of the experimental ingots is shown in Fig. 3.

Seven molds of the new design were cast and tested at the works. Steel for pipes made in a 75-ton open-hearth furnace with chrome-magnesite roofs was poured into the molds. The steel was bottom-poured through a magnesite nozzle of 40-mm diameter. After pouring, the ingots were put in storage and then rolled into tube billets of 180-mm diameter and inspected in groups as poured.

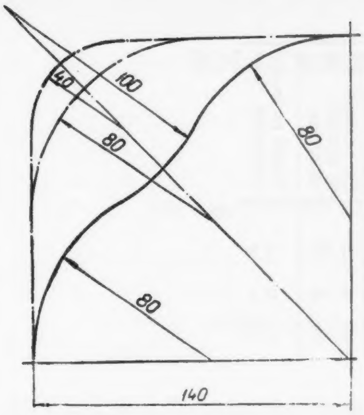


Fig. 2. Corners in the molds of the ordinary and the new designs.

During the period of investigation, the total faulty production of tube billets approached 7.4% of inspected material. The billets were rejected mainly on account of fissures and cracks. In order to test the resistance of the new design molds against the formation of hot cracks on the ingots, the molds were placed on one bottom plate together with ordinary molds (two at a time) and were teemed at a higher rate (400 mm/min). In nine out of ten test pourings the ordinary ingots had longitudinal corner cracks and lateral cracks; the new shape ingots had longitudinal cracks in one case only.

Temperature measurements on heating of ordinary and new ingots showed that the corners of the ordinary ingots were heated up considerably faster than the sides (the average temperature difference between corners and sides was 125°C) while the temperature difference in the case of the new design ingots did not exceed 40°C.

There was a smaller sideways spreading of test ingots in the first passes on rolling in the blooming mill.

With a view to determining the comparative percentage of rejected tube billets, the bottom plate with the test molds was placed second, third and fourth in order of pouring (there were five bottom plates per heat). Then the bottom plate with the molds of the new design and the neighboring bottom plates were inspected separately. The results of the inspection of 23 heats are shown in the table. The amount of rejected billets from the ingots, cast in the new test molds is 2-2.5 times lower than from the ordinary ingots. Bottom trimming of the new shape ingots is half that of the ordinary ones (1.5% instead of 3%). At the same time no sign of shrinkage was detected by macroinspection.

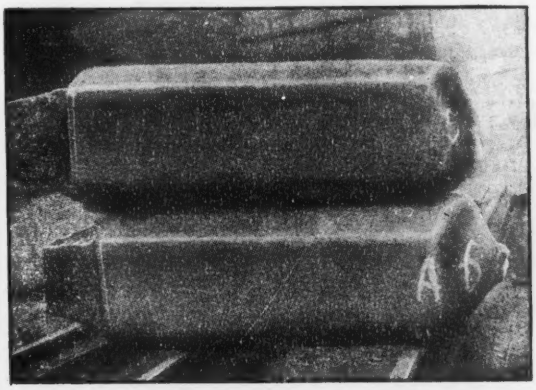


Fig. 3. Experimental ingots.

The durability of the experimental molds is practically the same as of the ordinary ones. The molds became unserviceable on account of a network of erosion, mainly on the walls in the lower third of the mold.

The investigation shows substantial advantages in the molds of the new design.

Comparison of Faulty Billets from Ingots Cast in Molds of Old and New Design

Sequence of bottom plates in the order of pouring	Steel inspected, t	Rejected material, t			Total rejected material, %
		On account of cracks	On account of fissures	On account of macro-structure	
Bottom plates, teemed prior to the test plates	286	5.66	8.47	0.80	5.0
Bottom plates with the test molds	299	2.13	3.78	0.49	2.1
Bottom plates, teemed after the test plates	299	4.50	6.37	0.84	3.9

ANOTHER OXYGEN PLANT

G. Romanov

A high-output oxygen plant has been put into operation at the Cheliabinsk Metallurgical Works. The plant consists of several units: an air distributor, oxygen compressor station, oxygen holder, circulation pump station, and cooling tower, as well as the so-called distant air intake which is located in the "green area" of the works. With the experience of other oxygen plants taken fully into account, a considerable part of the equipment has been modernized.

In the plant all processes are fully automatic, from the air intake to the delivery of oxygen into the production shops. The rated output of the plant is 10,800 m^3/hr.

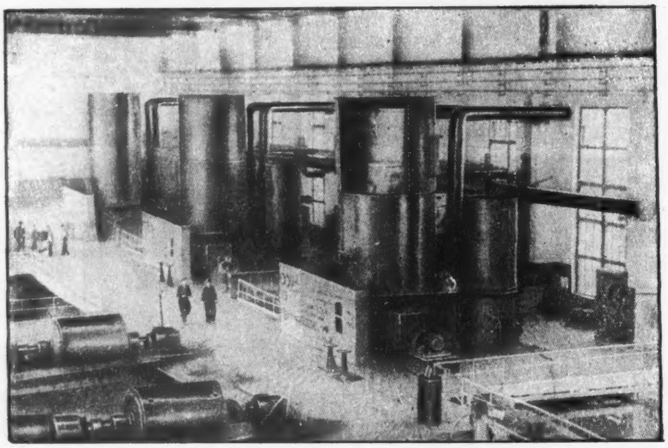


Photo by V. Aleksandrov.

The Figure shows the air distributing unit of the new oxygen plant.

ROLLED STEEL AND TUBE PRODUCTION

AN IMPROVED ROLL DESIGN FOR ROLLING STRIPS

G. K. Eremin

(Chief Roll Designer for the Sulinsk Metallurgical Plant)

For a long time the output in rolling 20 × 10, 22 × 10, and 22 × 13 mmstrips was low in the light-section mill of the Sulinsk Metallurgical Plant rolling shop. A considerable proportion of the output was of second quality and the waste – due to obliquity exceeding 3° – was high. The reason was the unsatisfactory design of the finishing line rolls, the passes of which are shown in Fig. 1.

The shortcomings of this roll set were as follows: pass 1 had the shape of a square outline by straight lines; in the No. 3 pass preceding the edging pass, the short edges of the rod became bulged; while two long sides remained straight, both side walls of the edging pass 4 had a considerable taper (5°).

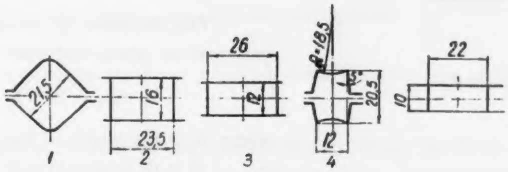


Fig. 1. The original roll passes set for rolling the rod.

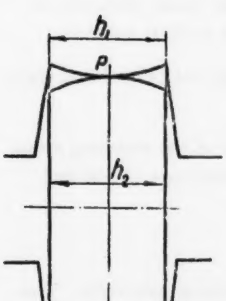


Fig. 2. Position of the piece with bulged edges in the edging pass.

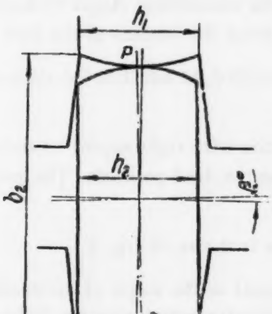


Fig. 3. Tilting of the piece in the edging pass.

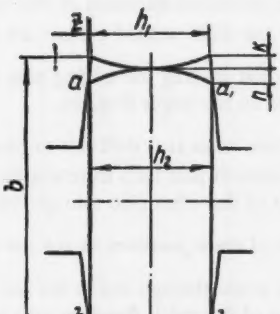


Fig. 4. Wide rod entering the edging pass.

If the condition $h_1 = h_2$ (h_2 is the width of the rod entering the pass and h_1 is the width of the edging pass) is fulfilled, then the rod with bulged edges will touch the edging pass walls only at two points, p and p_1 (Fig. 2). The rod entering the pass has no contact with its walls since they are inclined at an angle of 5° to the vertical.

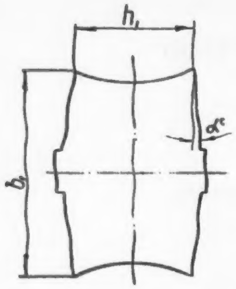


Fig. 5. Rod with bulged edges and projecting side strip after leaving the edging pass.

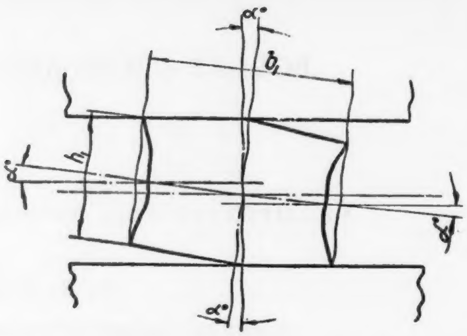


Fig. 6. Rolling of the rod on smooth rolls.

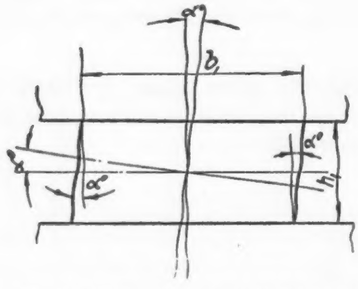


Fig. 7. Obliquity of the finished section.

The piece not supported by walls tilts to one or the other side until it makes contact with the sides of the grooves ("tilting" of the rod in the edging pass) as shown in Fig. 3.

In order to prevent this from happening, the width of the rod was made slightly larger than the width of the bottom of the edging pass. In this case the rod touches the pass walls at points a, a_1; and b, b_1 which retain it in an upright position (Fig. 4).

The thickness h_2 of the rod entering the edging pass can be calculated from the formula:

$$h_2 = h_1 + 2z = h_1 + 2\,(k + n)\ \mathrm{tg}\ \alpha,$$

where h_1 is the width of the bottom of the edging pass;
k is the convexity of the edging pass bottom;
n is the convexity of the edges of the rod entering the pass;
α is the angle of inclination of the pass side walls.

The increased thickness of the rod makes the succeeding stages of shaping the piece more difficult; in particular, no right-angled corners are obtained since the corners of the pass take no part in their shaping.

The rod leaving the edging pass is obtuse angled; in addition the sides of the section are bulged and show projections on the edges (Fig. 5).

In both cases it is difficult to obtain a section with right angles on the smooth rolls of the finishing stand since the smooth pass fails to retain the rod in the required position. The piece turns sometimes to one and sometimes to the other side through the angle α.

One of these positions of the rod in the pass is shown in Fig. 6.

The angle through which the rod turns is equal to the angle of inclination of the edging pass sides. The inclination of the rod is fixed by rolls and the obliquity remains on the finished section after the rolling is completed (Fig. 7).

In order to eliminate all these shortcomings a new set of passes (Fig. 8) was designed for this rod; in the edging pass the inclination of the side walls was reduced from 5 to 3°; recesses were made in the smooth rolls preceding the edging pass and the straight sides of the square pass were replaced by a bulged shape (No. 1 pass). The concavity of sides has a radius R and depth δ.

$$\delta = \frac{\Delta b_2 + \Delta b_3 + 2k}{2}.$$

where Δb_2 is the widening of the rod in the No. 2 pass;

Δb_3 is the widening of the rod in the No. 3 pass;

k is the depth of the bulged portion of the bottom of the edging pass.

The side of the square pass is found from the following formula:

$$a = \frac{b + hc + (b_2 - b_1)(1 - c^2)}{1 + c},$$

where b is the width of the finished section;

h is the thickness of the finished section;

c is the mean coefficient of the widening of the rod;

b_1 is the height of the edging pass, or the height of the rod leaving the edging pass;

b_2 is the width of the piece entering the edging pass;

$(b_2 - b_1)$ is the reduction in the edging pass.

In rolling with the new set of rolls the rod leaving the No. 3 pass (preceding the edging pass) has concave sides.

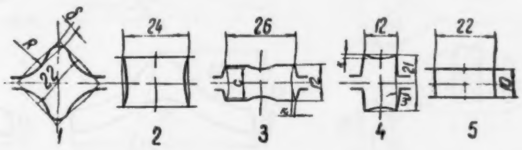


Fig. 8. New design of passes for rolling rods.

The bottom and top faces of the rod have a double concavity the size of which is equal to the widening of the metal in the edging pass.

The new method of rolling created normal conditions for rolling the 20 × 10, 22 × 10 and 22 × 13 mm strips and enabled loop rolling to be introduced on the finishing line. In addition, the second quality rods due to obliquity have been completely eliminated and the output of the mill considerably increased.

RATIONAL ROLL DESIGN FOR 280 MILL

Chief Roll Designer I. M. Kochetov *and Head of the Rolling Shop* V. F. Agarkov

(Saldinsk Metallurgical Plant)

The 280 mill (Fig. 1) has two lines: a roughing line consisting of two three-high 375 stands and one two-high 375 stand and a finishing line which comprises five two-high reversible 280 stands. The mill produces

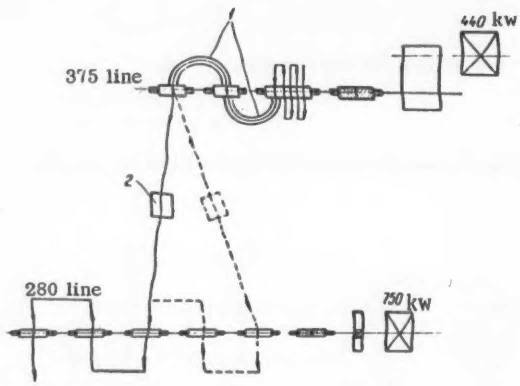


Fig. 1. Schematic diagram showing the layout of the 280 mill.
1) Repeaters; 2) transfer table.

mainly rounds, flats, channels, angles, and a window-frame section. Rounds, flats, and channels are rolled on the 375 stands with two repeaters and on the finishing 280 stands with two to three repeaters. The 35 × × 35 × 4-5, 45 × 30 × 4 mm angles and the No. 8 (55 × 25 × 4 mm) window-frame section are rolled on the finishing line of the mill by hand, without the use of repeaters. A considerable number of operators used to be engaged in this work.

Earlier, the 35 × 35 and 45 × 30 mm angles were rolled from an 80 mm square billet in seven passes on the roughing line and five passes on the finishing line (Fig. 1, dotted line); the No. 8 section was rolled in five and five passes on each line respectively.

The redesigning of the rolls enabled the two passes in the first and second stands of the finishing line to be eliminated. From the 375 line the piece was now fed into the third stand of the 280 finishing line (Fig. 1, continuous line).

The modification of the roll passes for rolling these sections are shown in Figs. 2, 3 and 4.

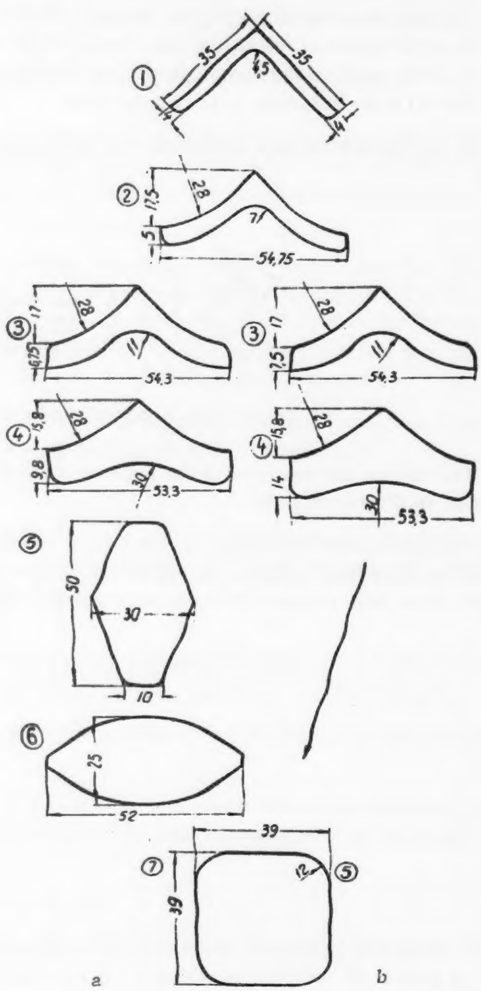


Fig. 2. Passes for rolling 35 × 35 × 4 mm angles.
a) Original passes; b) new passes.

The new roll design enabled:

1) an increase in the mean coefficient of reduction in rolling the 35 × 35 × 4 mm angle from 1.32 to 1.52 and in rolling No. 8 section from 1.32 to 1.42;

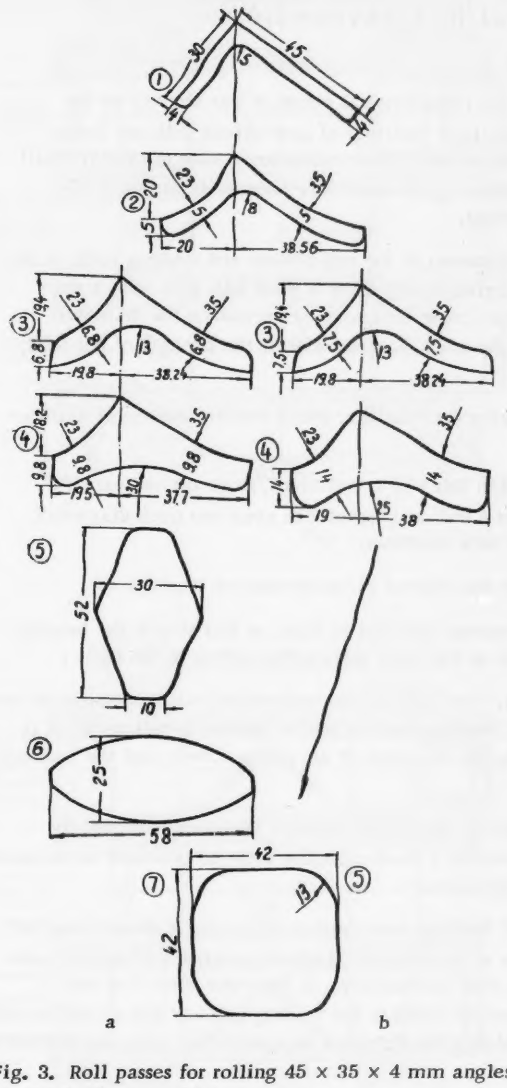


Fig. 3. *Roll passes for rolling 45 × 35 × 4 mm angles.*
a) Original passes; b) new passes.

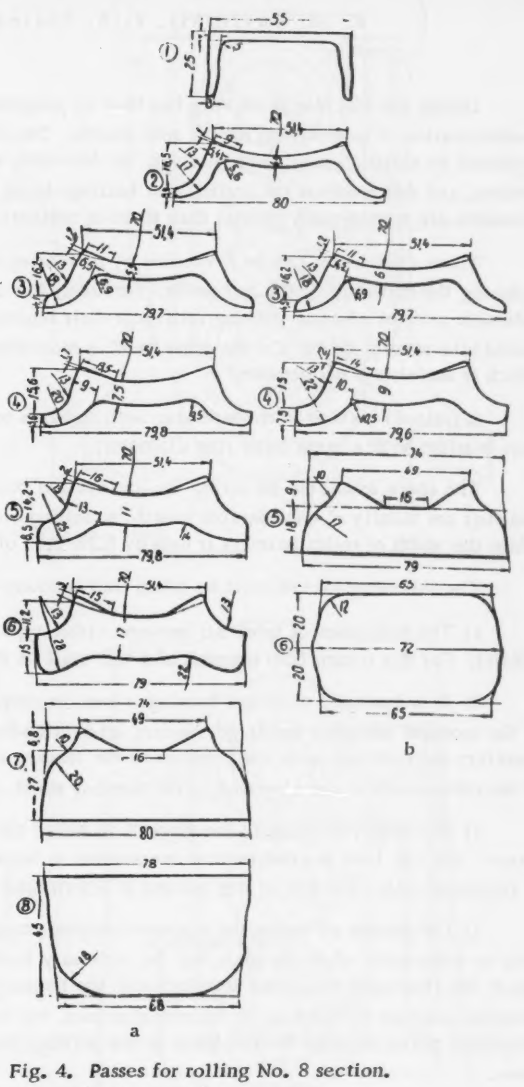


Fig. 4. *Passes for rolling No. 8 section.*
a) Original passes; b) new passes.

2) three operators to be relieved for use in other work;

3) the output of the mill in rolling these sections to be increased by 15-20%;

4) *the quality of output to be increased (second quality output was reduced by 10%);*

5) a reduction of power consumption on the finishing line;

6) the roller wear in the finishing line to be reduced.

At first it appeared that the increase of reduction, especially in the third stand of the 280 line, would lead to a rapid wear of the rolls; however, the reverse was the case—*the life of the rolls in the finished line increased* two to three times. After redesigning the rolls the temperature of the piece being rolled increased, which almost *entirely eliminated breakages of the rolls.*

MODERNIZATION OF GEAR-DRIVEN ROLLING MILLS

S. M. Ruvinskii, I. S. Starets and D. I. Shuliatskii

During the last few years work has been in progress at the metallurgical plants of this country on the modernization of gear-driven rolling mill stands. The babbit-lined bearings of gear-driven rolls are being replaced by antifriction (roller) bearings. In this work, stands are sometimes encountered with relatively small pinions, and this prevents the antifriction bearings being mounted in the usual way because their radial dimensions are considerably greater than those of ordinary bearings.

These difficulties can be overcome by increasing the diameter of the roll pinions and working rolls, or by reducing the diameter of the roll necks. However, this is not always easy since a plant has, as a rule, a considerable number of spare working rolls; and their replacement, made necessary by increasing the diameter, would take several years. On the other hand, a reduction of the neck diameter affects the strength of the rolls, which is not always permissible.

A rational method of modernizing such stands is to stagger the bearings, which enables the use of antifriction bearings with a large outer ring diameter.

The space available for fixing the antifriction bearings in this way is sufficient, since the ordinary bearings are usually of considerable length, equalling one and a half and sometimes even two neck diameters, while the width of roller bearings is usually 0.25-0.35 of the neck diameter.

The following factors must be taken into account when this method of modernization is adopted:

1) The difference of intervals between either type of bearing must not be high, as this affects the bending stresses. For this reason both supports of a roll must be moved to the same side, to the left or to the right .

2) It is desirable to mount bearings closer to couplings, especially at the main-drive side; otherwise the arm of the moment can grow too large, causing an overloading of bearings and excessive stresses in roll necks; it is therefore desirable to move the bearings of the leading roll in the direction of the prime mover, and the bearings of the driven rolls in the direction of the working stand .

3) The length of necks in the supports in which the bearings are moved outward becomes considerably larger: this can lead to considerable stresses due to bending forces; a thicker portion must be provided on the necks of staggered rolls: the size of this portion is determined by calculation .

4) The process of boring the supports carrying staggered bearings is somewhat difficult. It should therefore only be undertaken when the plant has the necessary facilities at its disposal. Supports consisting of several parts which are internally machined together with the housing are most satisfactory. In the cases when it is not possible to set up the housing on the machine tool, the supports are fixed to the housing grooves and welded to the temporary plates in order to bore them in one setting, maintaining the distances to a previously selected reference plane.

Some typical examples of modernized gear-driven stands are given in the following: Figure 1 shows the stands of a 270 wire mill; the root-circle diameter of the pinions was 258 mm and the neck diameter was 160 mm. This ratio of diameters prevented the roller bearing from being fitted in the usual way; even bearings of the lightest type for a 160-mm shaft have an outside diameter of 270 mm which is more than the root-circle diameter of the pinions.

Modernization involved the staggering of the bearings. As a result, the distance between the supports was only increased from 670 to 690 mm, which was considered safe. The condition of the right-hand neck was less favorable; here the distance to the center of the support lengthened from 120 to 195 mm which increased the loading by 62%. In order to avoid dangerous, excessive stresses, the free portion of the neck was increased in diameter to 190 mm, as a result of which the stress in the critical section became lower than it was before the modification.

The cylindrical roller bearings without shoulders on the inner ring facilitate assembly and enable the self-alignment of rolls in an axial direction; this is very important in the case of staggered bearings which make an accurate meshing of teeth difficult.

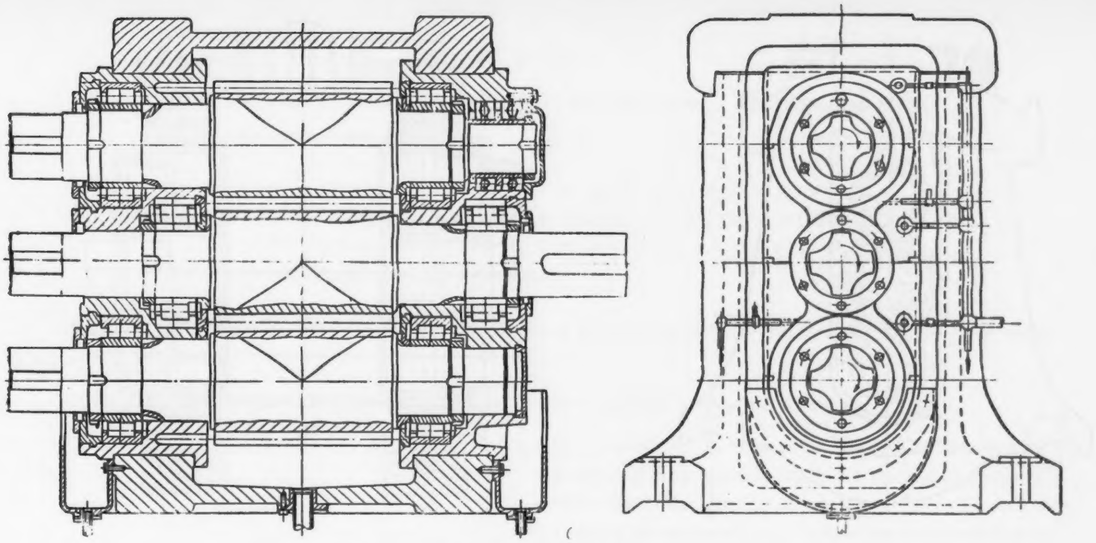


Fig. 1. Gear-driven stand of 270 wire mill.

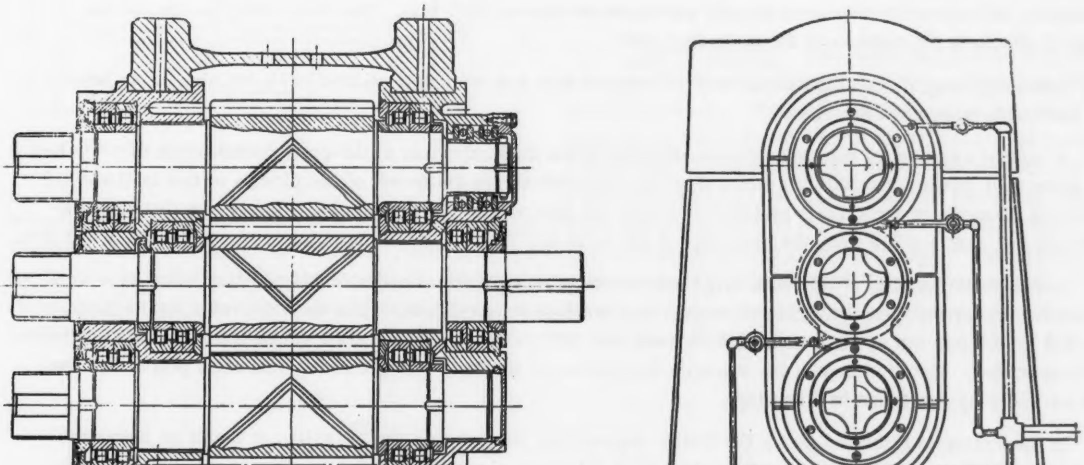


Fig. 2. Gear-driven stand of 300 merchant-section mill.

All three staggered rolls are prevented from moving in an axial direction by a single support consisting of two thrust ball bearings mounted on the top roll.

Since the rolls carry no axial forces, and the axial shocks due to incorrect machining of teeth are damped to a considerable extent during the axial movements within the clearance, i.e., during the "floating" of rolls, the thrust bearing need not be large and can be mounted in easily accessible places. The bearings are lubricated from a central circulation system with a liquid lubricant.

One or two bearings can be mounted in each support according to the need. This is shown in Fig. 2 which illustrates the modernization of a gear-driven stand of a 300 merchant-section mill. Here each neck is supported

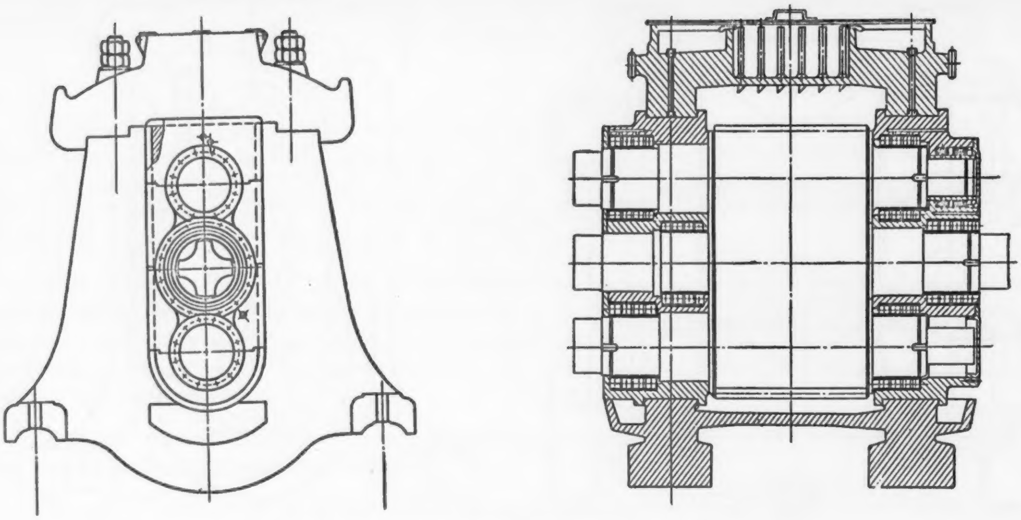


Fig. 3. Gear-driven stand of a three-high plate mill.

in two roller bearings; each pair of bearings is selected according to their internal initial radial clearances; the permissible difference of clearances in each pair must not exceed 0.03 mm. The lubricating system and the design of chocks is the same as in the preceding unit.

The staggering of bearings can be used not only in wire and merchant-section mills but also in big heavy-duty units with large neck diameters.

A typical example of this type of reconstruction is the modernization of the gear-driven stands of a three-high plate mill driven by a 3000 kw motor (Fig. 3). The root-circle diameters of the pinions of this mill are different (the diameter of the middle pinion is 638 mm and that of the driven pinions 798 mm); the distances between the roll axes are 718 mm; the diameter of all necks is 520 mm and the diameter of wobbler parts is 458 mm.

In the modernization of this unit, large-size double-row cylindrical roller bearings without shoulders were adopted for supporting the rolls. In each support two bearings were mounted. The neck diameter was reduced from 520 to 508 mm but as the overhand decreased from 320 to 228 mm this had no effect upon the mill's reliability in operation. The diameter of the strengthened portion of the neck was 610 mm. The axial position of the neck was fixed by two thrust ball bearings.

In the examples discussed above the fixing supports are mounted on the top rolls. It would be better to mount them on the middle (driving) roll as this would eliminate the axial play in the driving roll and the main drive coupling, but this is difficult to achieve. The test carried out on a number of units showed that it is of little importance on which of the rolls the fixing thrust bearings are mounted, since in the gear-driven stands with ordinary bearings the axial play is due to the considerable clearances between the end faces of pinions and bearings. In operation these clearances are continuously increased and sometimes reach the values of 5-6 mm.

In mounting thrust ball bearings the axial clearance is adjusted within 0.2-0.4 mm and does not change for a considerable period of time. The positioning of the thrust unit on the drive side of the top roll makes it most easily accessible for regular inspection or regulation.

The design life of radial bearings in the gear-driven stands is very high indeed and reaches 25,000 to 80,000 hr.

The recommended modernization method of gear-driven stands can also be used in the modernization of the heavy equipment of rolling mills and of other machines.

THE MANUFACTURE OF CHROMIUM-PLATED GUIDES FOR ROLLING MILLS

M. A. Tylkin

(Head of the Thermal-Treatment Department of the Dzerzhinsk Plant Machine Shop)

In recent years roller type guides have gained considerably in popularity since, as experience shows, they facilitate the feed of the rod into the pass and ensure a better surface quality of the rolled material.

Owing to their short life, the guides are a bottleneck in the operation of a wire mill. They must often by replaced every shift.

At the Dzerzhinsk plant heat treated chromium plated guides were put into use.

The finishing guides of the 260 and 330 mill (Fig. 1) are made of St. 3 or 20Kh steel. The blanks are normalized as follows: they are heated to 880-900°C and maintained at this temperature for 1-3 hours (according to composition), with subsequent cooling in air. The normalized blanks are milled and planed, whereupon they undergo heat treatment which consists of the following operations: case hardening, normalization, hardening, and tempering (Fig. 2).

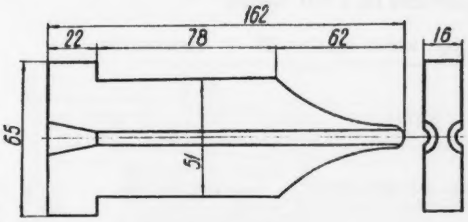


Fig. 1. Finishing guides of wire mill.

Case Hardening. The blanks are placed into a metal box filled with a suitable carbonizer (90% charcoal and 10% soda ash). The box is then hermetically sealed with a cover, coated with fireclay and placed in the furnace. The heating to case hardening temperature (900-920°C) lasted 4 hours. As a result of prolonged heating at this temperature carbon penetrates into the top layer of the metal, forming a eutectic layer of a certain thickness. The duration of heating is determined by the need to obtain a definite thickness of the hard layer. The guides for the 260 mill had to be maintained at the carbonization temperature for 12-14 hours. This ensures a depth of carbon penetration into the surface layer of 1.2-1.5 mm; for the guides for the 330 mill the corresponding time was 15-18 hours. This ensures a carbonized layer thickness of 1.5-1.8 mm.

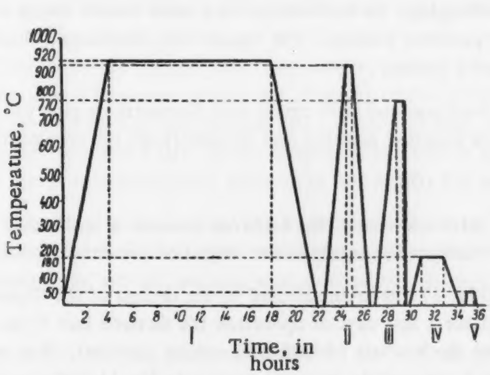


Fig. 2. The graph of heat treatment and chromium plating of guides.

After carbonization the boxes with guides are cooled in air.

Normalization. In order to improve core grain the guides were heated in the furnace to 880-900°C, kept at this temperature for one hour and subsequently cooled in air.

Hardening. In order to produce a surface of great hardness and strength without affecting the properties of the core, each guide is heated in the furnace to 760-780°C, maintained at this temperature for half an hour and cooled in cold water (guides are immersed into water faces first).

The hardness of the groove after hardening should not be below 60-62 R_C.

Tempering. In order to remove the stresses produced in the component during hardening, the guide is heated to 160-180°C, maintained at this temperature for two hours and cooled in air. After tempering, the hardness of the working groove should not be below 60 R_C.

The heat treated components are sand-blasted and then chromium plated. Prior to chromium plating the working groove of the guides is polished on a felt wheel coated with 100 grit emery dust and degreased in a 5% sulfuric acid.

The guides are then clamped into special textolite devices in which both grooves correspond to those of the guides, whereupon they are hung on the rods in the plating bath. The bath contains the following solution: 240 g of chromium anhydride in one liter of distilled water and 2 g of sulfuric acid in one liter of distilled water. The temperature of the bath is 50°C, voltage 8-10 v, and current density 50 amp/dm^2. A chromium layer 0.03-0.05 mm thick is formed in 30 minutes. After plating the working groove is polished on a felt wheel.

This method of manufacturing the guides provides for a long life, which reaches 20 or more shifts.

RECONSTRUCTION OF A CONTINUOUS FURNACE

Head of the Design Office P. K. Kuznetsov

Furnace Foreman of the Rolling Shop, E. Ia. Klassen

(Gur'ev Metallurgical Plant)

Until 1954, two continuous furnaces with monolithic hearths in which ingots and billets were heated only from above, were used to supply rolling mills. In order to obtain a uniform heating of ingots they were turned in the furnace by hand before discharging. In the furnace they were moved along metal beams sunk into the floor by means of electric motor-powered pushers. The ingots were discharged through a side door along a water-cooled cast iron plate by means of a pusher.

The furnaces were heated from one end with small-size Kuznetsk-region gas coal, burned on inclined grates. Two medium-pressure fans supplied primary and secondary air for combustion.

The output of a furnace was 7.9 t/hr.

When the output of rolling mills increased, the furnaces became a bottleneck in the shop. It became necessary either to build a new furnace or to increase the output of the existing units.

In 1954 the No. 2 furnace (Fig. 1) was reconstructed to the design of the Gipromez Sverdlov branch, providing for an output of 20 t/hr. After a few days of operation the furnace had to be altered since the horizontal grate proved unsuitable for burning the low-ash (6-10% ash) caking gas coal. The ash left after the combustion of this coal was in a pasty state and covered the grate, completely blocking the supply of air into the furnace. The operation of the furnace was unsatisfactory. In addition, the cleaning of the grate was difficult. The rocking fire bars became clogged with slag and ceased to operate.

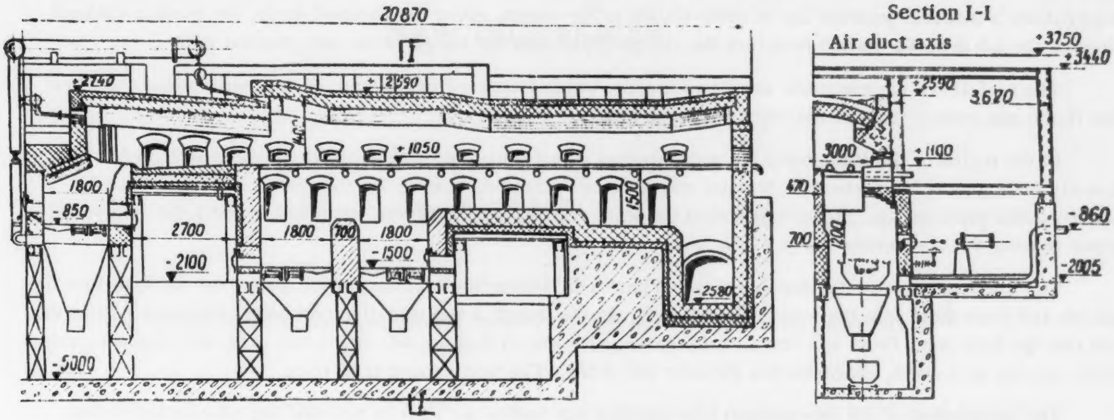


Fig. 1. Continuous furnace for heating ingots, before reconstruction.

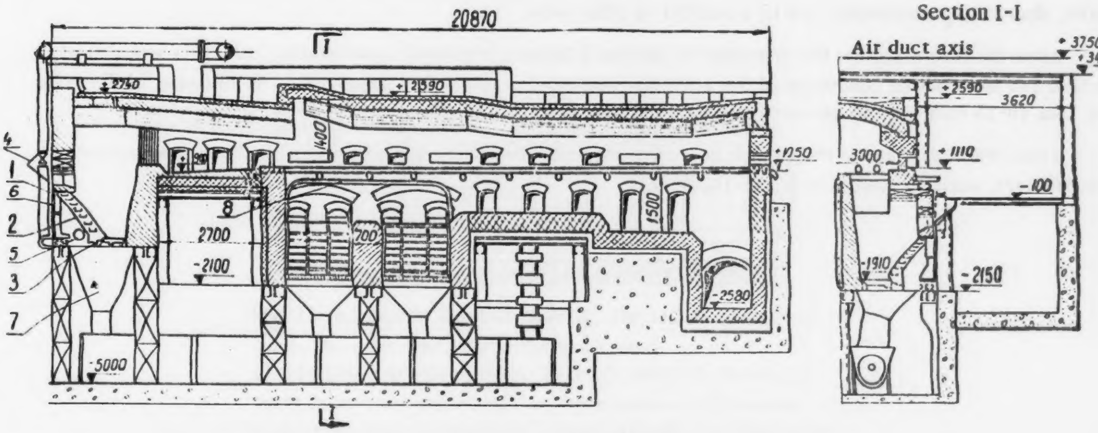


Fig. 2. Continuous furnace for heating ingots, after reconstruction.

Before cleaning the grate (for economy reasons) the combustion process was intensified and a red-hot layer of coal formed on top of the ash. Therefore, during the cleaning of the grates the rocker bars had to be lifted and the coal layer fell almost completely into the ash pits. After cleaning, the grate was bare and the furnace temperature sharply dropped; one and a half to two hours were needed to get the grate fully into operation. The coal consumption per ton of rolled material went up to 209 kg/t.

The secondary air which, according to the design, had to be supplied above the side grates, proved unnecessary, as it only cooled the metal in the furnace; the working conditions of the furnace personnel were difficult, especially as far as the stokers and ash-pit workers were concerned. It took nearly 8 hours to clean the grate.

In response to a suggestion submitted by a group of innovators of the shop, the No. 2 furnace was reconstructed (Fig. 2).

The horizontal grate was replaced by a grate of combined design consisting of an inclined section (inclined at 45°) and a horizontal section.

The frame (1) has air-tight doors for cleaning, and supports one end of the inclined grate (2); the bottom end of the grate rests on the cross beam (3); coal is filled through the loading doors (4). The air is fed through the duct (5) from a medium-pressure fan of 8000-10,000 m^3/hr output, under the inclined grate; the grate is cleaned through the ash doors (6); the ash falls into the ash pit (7) and into the trough of the ash-disposal unit.

The roof above the end grate was lifted higher, which increased the volume of the combustion chamber; the flame was moved close to the ingot discharge chute.

In the region of the side grates the water-cooled transversal pipes rested on the partition wall which was quickly destroyed by heat when the furnace was put into operation, causing the transversal and longitudinal water-cooled pipes to sag. In the new design the water-cooled support (8) was provided to carry the transversal pipes crossing the combustion space of the side grates.

The ash disposal from the furnace was mechanized. During the cleaning of the grates the ash falls into the ash pit and from there into the water-filled trough. In the trough a 400 mm diameter worm conveyor carries the ash into the feed box. From this box a chain-bucket elevator carries the ash into a car. The ash-disposal equipment consists of a worm, chain-bucket elevator and drive. The worm speed is 20 rpm.

The installation of the ash-disposal unit enabled one worker per shift to be freed and eliminated almost completely the heavy work of ash-pit operators.

Furthermore, in the discharge section of the furnace two additional water-cooled hammers were installed to facilitate the correct movement of metal along the slide tubes. Alterations were also made in the design of chutes, door lifting mechanism and in a number of other units.

After the reconstruction the operation of the No. 2 furnace improved considerably, its output was almost doubled and the working conditions of the attendant personnel considerably improved. The cleaning of the grate now lasts 10-15 minutes and has no effect upon the operation of the furnace.

There was a substantial drop in the coal consumption. Before the reconstruction the coal consumption was 194-209 kg/t, while at present it is 155-165 kg/t.

NEW TECHNIQUES

METHODS OF WORK EMPLOYED BY STOCK YARD FOREMAN A. N. ZHERNOVOI

K. P. Murzov

(Assistant Head of the Open Hearth Furnace Shop at the Stalin Metallurgical Plant)

The stock yard of the open hearth furnace shop, which runs parallel to the main building of the shop, is a covered building 152 m long and 30.5 m wide (measured on the column axes). It is equipped with two magnet-grab cranes of 10/10 ton capacity and two magnet cranes of 15 ton capacity each. The capacity of the grab is 2 m^3 and the power of the magnet is 11 kw. The stock yard has four railroad tracks: two are intended for unloading the cars and two for loading charging buggies (Fig. 1).

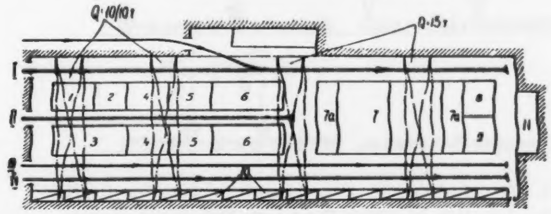


Fig. 1. Schematic diagram showing the layout of railroad tracks, bunkers, and bins in the stock yard.
1) Magnesite powder bunker; 2) rough dolomite bunker; 3) burnt dolomite bunker; 4) limestone bunkers; 5) bauxite bunkers; 6) iron ore bunkers; 7) iron scrap pit; 7a) fine scrap; 8) ferromanganese bunker; 9) 12% ferrosilicon bunker; 10) ferroalloy bins; 11) office; I) iron scrap unloading track; II) bulk (loose) material unloading track; III) bulk material loading into charging buggies; IV) track used for unloading ferroalloys, fine scrap, and half-filled ingots.

Charging materials are stored in special 4-5 m deep bunkers and bins.

The loading frontage for handling bulk materials takes 10 two-axle buggies on one track. Two charging-buggy trains, each consisting of 20 two-axle buggies are available for the transportation of bulk materials.

Metal scrap is transported in 0.91 m^3 capacity charging boxes (22 four-axle buggies) and in 0.56 m^3 capacity charging boxes (18 four-axle buggies); 15 buggies with 0.91 m^3 capacity charging boxes are allocated to the transportation of scrap from the finishing department to the open hearth furnaces (400-450 t/24 hours), the remainder is used for the transportation of scrap from the stock yard (600-700 t/24 hours).

The highly efficient operation of the stock yard depends on:

a) correct placing of the rolling stock in loading and unloading;

b) effective use of cranes in loading and unloading;

c) skill of crane drivers and the methods they use;

d) correct disposition of stock yard operators and organization of their work.

Stock yard foreman A. N. Zhernovoi bases the disposition of rolling stock on the scheduled shift output, location of scrap and its arrival for unloading, and on the possibilities of the use of cranes. Quick loading of metal scrap into the charging boxes is achieved when idle runs of cranes are avoided and the driver concentrates his attention on the work of the magnet; the bridge is used only to move the crane from one buggy to another.

Figure 2a shows the order in which the empty buggies are placed for loading and the method of employing cranes in this work (Fig. 2b). When the rolling stock is placed according to this plan, the cranes are used inefficiently. They obstruct one another with the result that the loading time is increased.

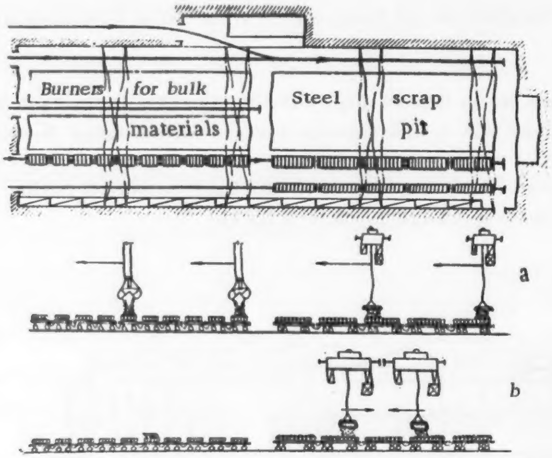


Fig. 2. The disposition of buggy trains during the loading and the direction in which the travelling cranes operate. *a) Method adopted by A. N. Zhernovoi; b) unsatisfactory* organization of the work of travelling cranes.

A. N. Zhernovoi arranges the operation of travelling cranes during the loading in such a manner that at certain times both cranes are engaged in loading the same train and then move on to the next train. This considerably reduces the shunting needed to replace a full by an empty train. Other stock yard foremen use the cranes for the simultaneous loading of two trains, as a result of which some cranes have idle periods, the time taken for loading is increased and the shunting made more difficult.

With proper care the crane driver can load metal scrap so that it remains in the charging boxes well within the loading line and at the same time frees loading workers for use in other jobs. Foreman A. N. Zhernovoi systematically introduces into the organization of the work of crane drivers and loading workers advanced, highly effective methods. For example, during his shift the crane driver only partially loads some charging boxes with scrap and goes on to fill other boxes. After the workers have finished levelling the scrap in the first charging boxes, the cranes complete their loading, while the workers are levelling the scrap in the next section of boxes; which procedure is continued until the whole train is loaded.

After the light scrap has been loaded into the charging boxes, heavier off-cuts are added until each buggy has reached the fixed average weight of 18-20 tons.

A continuous study of advanced methods and their introduction into the work of stock yard workers enabled the best way of organizing the production processes to be found, and also the best methods of preparing the metal charge and its loading into the charging boxes, the elimination of stoppages and the considerable reduction of the charging time of open hearth furnaces.

By organizing the work in this way comrade Zhernovoi reduced the time taken for loading one train by 30 minutes and saved 2 hours during a shift, which is the time needed for loading one train with metal scrap (90-100 tons). Other foremen often engage workers in various jobs of secondary importance (which comrade Zhernovoi often performs during the breaks in operation) and this has an adverse effect upon the productivity of workers and the work of magnet-grab cranes.

Time and motion studies carried out in the stock yard show that the time required for loading one crane (20 buggies) with bulk materials (200-230 tons) was 1 hour 20 minutes in the shift of comrade Zhernovoi, while in other shifts it was 1 hour 50 minutes-2 hours. The Zhernovoi method requires 1 hour 40 minutes-1 hour 50 minutes for loading one train (five 4-axle buggies) with metal scrap. (The replacement of a full by an empty train takes 15 minutes.)

HARDWARE PRODUCTION

THE MANUFACTURE OF THIN WIRE BY COLD ROLLING

I. S. Pobedin, V. I. Bairakov, M. G. Uglov and V. G. Drozd

At present wire of 5-6 mm diameter is produced by hot rolling while wire of smaller diameter is manufactured exclusively by drawing. However, this method has a number of disadvantages: reductions are small and limited by the strength of the leading end of the wire, the drawing speeds of alloy steels and especially of alloys with special properties (of the Kh15N60 type) are limited and do not exceed 2.0-2.5 m/sec. An increase of the drawing speed is restricted by the strength of the dies, the generation of heat in the dies and the metal and also by the lack of a suitable lubricant. For these reasons there is a tendency to replace wire drawings by a more efficient and cheap cold rolling.

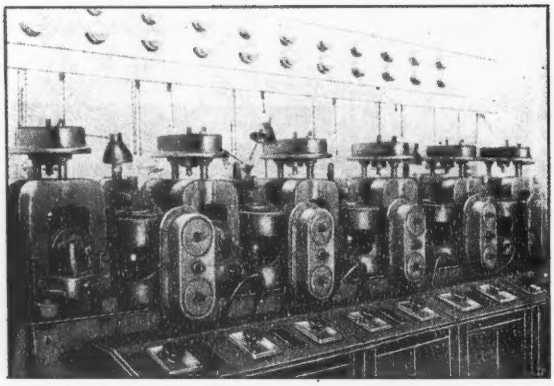


Fig. 1. 12-stand mill for the continuous rolling of thin wire.

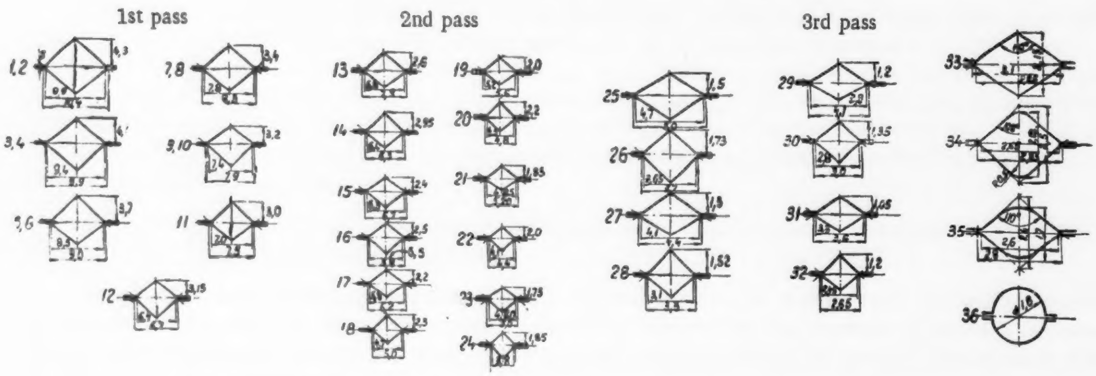


Fig. 2. Passes for cold rolling of 1.8 mm diameter wire from 8.0 mm wire rod. 1, 2, 3, etc. passes.

In 1951, the TsKBMM of the Central Scientific-Research Institute of the Ministry of Machine Construction designed and built a 12-stand mill for the continuous rolling of thin wire from special grade steels (Fig. 1).

The mill has working rolls of 140 mm diameter and 70 mm length (between the necks). Four grooves are cut on each roll. The rolling speed (in the last stand) is 2-3 m/sec, the power of a working stand is 7.9 hp, and the speed of a working stand motor is 1000-1750 rpm.

The mill is intended for the cold rolling of 1.5-2 mm wire from a 6-8 mm wire rod in three passes through the mill (i.e., in 36 roll passes) or for the hot rolling of 10-15 mm or 12-18 mm diameter rods into 6-8 mm diameter wire rod in one pass through the mill (i.e., in 12 roll passes).

The mill comprises two types of stands. All odd stands have vertical rolls, and all even stands have horizontal rolls. The working rolls of each of the stands are driven by individual electric motors through a two-stage reduction gear and a universal joint. The feeding device consists of a drum, straightening rolls and inlet guides.

The coils are wound by a reeler fitted with idle bending rolls. The coils are wound helically with turns of constant diameter, which is achieved by suitably tapering the drums.

One of the most difficult problems in developing this method of cold rolling wire was the design of passes.

Three systems were tried out during the development of the continuous rolling process: flat-flat (in smooth rolls), diamond-diamond, and diamond-square. Each system had three to four special intermediate passes (diamond or square) considerably rounded at the tops in order to obtain a round section (Fig. 2, grooves III, passes 34-35).

No satisfactory results were obtained with the flat-flat system because of the difficulties encountered in obtaining the section of the required dimensions. The diamond-diamond, and diamond-square systems enabled 1.8 mm diameter wire to be obtained from 8 mm wire rod in three passes through the mill without an intermediate annealing. A wire with a good quality surface was obtained.

The system of passes finally adopted was a mixed one: for the first pass through the mill the diamond-diamond, and for the second and third passes the diamond-square systems were chosen.

The diamond-square pass system enables square sections to be obtained in the intermediate passes. The shape of passes and the rolling data are given in Fig. 2 and in the Table.

Passes 1, 2, 3, 4, 5, 6, 7, 8, 9 and 10 are identical. In these passes the reduction is obtained by altering the gap between the rolls. In the diamond passes the ratio of the diagonals is successively increased and this lengthens the sides of small sections and creates better conditions for holding the wire in the guides.

In testing the mill the following measurements were made: the forces acting on the rolls, power consumption and rolling speeds in the individual stands in rolling wire from Kh15N60 alloy. The rolling forces and the power consumption are also given in the Table.

The absence of any regularity in the distribution of forces acting upon the rolls can be explained by the fact that in adjusting the mill the rolls are given a small initial pressure. In some stands this initial pressure is smaller than the rolling pressure, in others it is higher.

During the adjustment of the plant for rolling 1.8 mm wire from Kh15N60 alloy, it was found that guides 11 and 12 failed because of the extensive sticking of metal to the walls. In order to protect these guides it was decided to fit them with a 2848 cast iron lining with a white surface layer. With this lining it became possible to roll 700-800 kg of thin wire, whereas earlier the guides failed after rolling 12-15 kg. The new manufacturing method for 1.8 mm wire by continuous cold rolling, from steel with improved physical properties of the type Kh15N60, is more efficient than the drawing of wire, because it eliminates the need for intermediate annealings and the operations concerned with the preparation of the metal surface for drawing (pickling, washing, coating with lime).

The rolling speed is about 20-25% higher than the drawing speed.

It is advisable to have two mills for cold rolling thin wire. This considerably reduces the setting-up time, increases productivity and ensures a higher stability of the process.

On these mills it should be possible to roll 1.8 mm wire from 8 mm rod in one pass through each mill. The roll diameter of the second mill should be slightly smaller (about 100 mm) than that of the first unit, in order to obtain more accurate dimensions of the sections.

Compared with drawing, the efficiency of cold rolling can be improved still further by increasing the rolling speed. A preliminary test produced satisfactory results.

Roll Passes for Rolling 1.8 mm Wire from 8.0 mm Wire Rod

Calculated data									Actually measured					
No. of the pass	height of the wire rod h_0, mm	width of the rod b_0, mm	height of the pass h_k, mm	width of the groove b_k, mm	top angle of the groove β, deg.	gap between rolls S, mm	cross sectional area of the rod, F, mm^2	reduction	h_c, mm	b_0, mm	F, mm^2	λ	force acting upon the rolls P, t	rolling power N, kw
1st pass through the mill														
1	8.9	10.4	8.9	9.9	98	—	—	—	7.8	8.15	43.2	1.165	6.0	4.9
2	8.9	10.4	8.9	9.9	98	—	—	—	7.87	7.85	41.9	1.031	—	5.3
3	8.5	9.9	8.5	9.4	98	—	—	—	7.63	7.95	39.2	1.068	4.5	5.8
4	8.5	9.9	8.5	9.4	98	—	—	—	7.60	7.70	36.6	1.071	5.0	5.2
5	7.7	9.0	7.7	8.5	98	—	—	—	7.15	7.74	32.6	1.12	6.9	4.9
6	7.7	9.0	7.7	8.5	98	—	—	—	7.15	7.30	31.0	1.051	5.2	4.4
7	7.1	8.3	7.1	7.8	98	—	—	—	6.85	7.25	28.4	1.091	5.8	4.0
8	7.1	8.3	7.1	7.8	98	—	—	—	6.85	6.90	27.3	1.04	6.2	6.0
9	6.7	7.9	6.7	7.4	98	—	—	—	6.47	6.90	25.48	1.071	4.6	4.3
10	6.7	7.9	6.7	7.4	98	—	—	—	6.50	6.58	24.4	1.044	7.0	4.6
11	6.3	7.5	6.3	7.0	98	0.3	23.6	—	6.10	6.60	22.68	1.075	5.6	4.2
12	6.6	6.7	6.6	6.4	91	0.3	22.1	1.07	6.24	6.20	21.22	1.068	7.0	4.1
2nd pass through the mill														
13	5.5	7.3	5.5	6.8	105	0.3	20.0	1.10	5.26	6.57	18.08	1.175	3.8	2.8
14	6.2	6.3	6.2	6.0	91	0.3	19.5	1.03	6.0	5.36	17.25	1.048	2.9	1.9
15	5.1	6.7	5.1	6.3	105	0.3	17.0	1.14	4.8	6.04	16.28	1.06	3.6	3.5
16	5.3	5.8	5.3	5.5	91	0.3	15.2	1.12	5.28	5.0	15.01	1.085	3.7	3.4
17	4.7	6.2	4.7	5.8	105	0.3	14.5	1.05	4.47	5.50	13.6	1.104	3.2	2.5
18	4.9	5.0	4.9	4.7	91	0.3	12.25	1.16	4.64	4.70	12.33	1.103	2.5	2.4
19	4.3	5.6	4.3	5.2	105	0.3	12.0	1.03	4.10	4.85	11.43	1.079	3.1	1.5
20	4.7	4.8	4.7	4.5	91	0.3	11.25	1.07	4.43	4.22	10.72	1.066	1.8	2.2
21	4.0	5.2	4.0	4.85	105	0.3	10.8	1.04	3.75	4.45	9.66	1.110	3.9	2.2
22	4.3	4.4	4.3	4.1	91	0.3	9.6	1.13	4.02	3.88	9.15	1.056	1.9	2.5
23	3.8	5.0	3.8	4.60	105	0.3	9.3	1.03	3.62	4.17	8.71	1.051	2.8	2.5
24	4.0	4.1	4.0	3.8	91	0.3	8.2	1.13	3.67	3.69	8.05	1.082	2.9	2.4
3rd pass through the mill														
25	3.15	5.0	3.15	4.7	115	0.15	7.8	1.04	3.08	4.12	7.36	1.094	—	—
26	3.61	3.7	3.61	3.55	91	0.15	6.4	1.2	3.46	3.36	7.0	1.052	—	—
27	2.75	4.4	2.75	4.1	115	0.15	6.0	1.05	2.62	3.75	6.03	1.162	—	—
28	3.19	3.3	3.19	3.1	91	0.15	5.3	1.13	3.16	2.72	5.44	1.108	—	—
29	2.55	4.1	2.55	3.8	115	0.15	4.9	1.08	2.56	3.24	5.17	1.05	—	—
30	2.85	3.0	2.85	2.8	91	0.15	4.3	1.13	2.77	2.70	4.68	1.102	—	—
31	2.25	3.6	2.25	3.3	115	0.15	4.0	1.06	2.23	3.08	4.23	1.10	—	—
32	2.55	2.65	2.55	2.45	91	0.15	3.6	1.10	2.55	2.40	3.92	1.08	—	—
33	2.15	3.1	2.15	2.85	110	0.15	3.3	1.09	2.04	2.78	3.50	1.12	—	—
34	2.08	2.85	2.08	2.65	108	0.15	3.02	1.09	2.05	2.27	3.07	1.14	—	—
35	1.7	2.8	1.7	2.6	110	0.15	2.76	1.08	1.68	2.50	2.68	1.149	—	—
36	1.8 mm diameter circle		—	—	—	0.15	2.55	1.08	1.85	1.80	2.54	1.054	—	—

M. A. PAVLOV

Soviet metallurgy suffered a severe loss on January 10, 1958 when the renowned Soviet metallurgist, Academician M. A. Pavlov died at the age of 95.

Mikhail Aleksandrovich Pavlov was born on January 22, 1863 in the Transcaucasus near the city of Lenkoran, of a Cossack family. In 1880, after finishing the Baku Secondary School, M. A. Pavlov entered the St. Petersburg Mining Institute where he completed the course in 1885. The engineering activities of Pavlov began in small metallurgical plants in the Vyatka mining region (at first at Omutninsk and then at Klimkovsk). Here he showed himself a resourceful engineer and an acute investigator. He studied generator gas, blast furnaces with different blowers, etc.

In 1896, when he was already a mature engineer, M. A.Pavlov entered the Sulinsk Metallurgical Works where he ran blast furnace fusions with anthracite. This was antirely new activity, for there was no specialist in Russia acquainted with fusions with anthracite. On the basis of his careful studies of this subject, Pavlov made significant changes in the construction of the blast furnace and worked out new technological methods. This allowed him to obtain the best productive results.

The publication in the Mining Journal in 1894 of the paper "A Study of the Fusion Process in the Blast Furnace of the Klimkovsk Works" was particularly important for the theory of the blast-furnace process. This work showed that Pavlov had unusual ability as an investigator, and played a decisive part in the choice of Pavlov in 1900 as a professor in the Ekaterinoslav Higher Mining School without his defending a dissertation. In 1904 he was chosen as professor in the St. Petersburg (now Leningrad) Polytechnic Institute where he worked until 1941. In 1921 M. A. Pavlov also became professor in the Moscow Mining Academy and later the Moscow Steel Institute. In 1927 Pavlov was chosen a Corresponding Member and in 1932 an Active Member of the Academy of Sciences of the USSR.

The theoretical questions which Pavlov took up in his noted work "A Study of the Fusion Process in the Blast Furnace of the Klimkovsk Works" were among the most important for the theory of the blast process. This study showed the results of long and painstaking work (Pavlov himself ran the chemical analyses of the raw materials and the products), and several years were spent on it. In the study he used two blast furnaces with the same productivity, one using a hot blower, the other a cold.

In this classical work Pavlov analyzed the economics of the fuel for heating the blower and for the first time very clearly expressed the idea, "every factor which lessens the expenditure of heat in a blast furnace gives a greater fuel saving when the heat efficiency in the blast furnace is decreased." This idea M. A. Pavlov himself called the "Okkerman principle" since Okkerman came to the same theoretical conclusion when studying the effect of heating the blower on the running of the furnace. The idea of Pavlov, however, is more general than the statement of Okkerman, and actually it should be called the "M. A. Pavlov principle."

In the same notable work Pavlov solved another important problem of the theory of the blast process — the effect of direct reduction on expenditure of fuel. When he analyzed the results of the study of the Klimkovsk blast furnace he concluded that the theory of the "ideal course" developed by the famous French metallurgist Gruner was incorrect. By the "ideal course" Gruner implied a process in which all the iron was reduced indirectly (i.e., by carbon monoxide). Pavlov showed that such a course was "ideal" only in respect to expenditure of heat in the reduction of iron, but not in respect to the expenditure of carbon as the reducer. Noting that the indirect reduction of iron from the oxide required a great excess of carbon monoxide (and hence of carbon) while direct reduction was carried out with less carbon, Pavlov showed that a nearly ideal run of the furnace "most often

occurred with great expenditure of fuel," but "direct reduction does not produce the harm ascribed to it," if the heat required for it is compensated by raising the heating of the blower.

The theoretical work of M. A. Pavlov was completed by his great study "The Metallurgy of Cast Iron," which particularly showed his inherent ability for wide scientific generalization. The scholarly exposition of this book of his clear and complete theories of the blast furnace, which were recognized throughout the world, gave a deep analysis of the phenomena of the blast process. The book "The Metallurgy of Cast Iron" was expressed in the concise and clear language which was always characteristic of M. A. Pavlov. Some of the present metallurgists of the USSR were trained from this book. The book, "The Metallurgy of Cast Iron," first appeared in 1924, if we do not count the edition of 1910 in which questions of the plans and construction of blast furnaces were discussed, and the appearance of each new edition was an important event, since in each of them he summed up the latest advances in theory and practice of blast-furnace production. This work was translated into many languages and was highly valued by foreign scholars.

Pavlov's great services furnished the scientific basis for many metallurgical calculations. Here it is particularly important to mention his work in drawing up tables of the heat effects in chemical reactions and heat capacities which furnished information as to "how metallurgists should use them." These tables which were based on a critical study and evaluation of a great number of original physicochemical studies were of the greatest value to many metallurgists.

M. A. Pavlov worked out a strictly scientific method for calculating the charge of the blast furnace. He considered that the most important problem in this field was the physical condition of the slag, since on this depended the manufacture of cast iron of the required composition and the proper run of the furnace.

M. A. Pavlov's book "Calculations for Blast Furnace Charge" was first published in 1914 and appeared in seven editions, becoming an important aid for every Soviet blast-furnace operator. In this book the scholar gave a critical evaluation of methods of calculation used earlier and described his original ("rational") method of calculating the charge for different cases of practical blast-furnace production.

Pavlov's work devoted to plans for blast furnaces was very valuable. He studied the vast amount of material on the actual blast furnaces of the most important metallurgical regions of the world and established which furnaces gave the best technical and economic results under given conditions of fusion; he proposed that at definite diameter of the furnace there be a definite set of relations between the other parts of the design. The bood "Determination of the Dimensions of Blast Furnaces" was first published in 1910 and often reprinted in the USSR and abroad; it gave a method of calculating the design which always gave good results and this method continues to be used for starting projects in the USSR at the present time. In this and other works Pavlov advocated the most progressive technical methods for blast operation.

After the Great October Revolution, an expanded program of investigation of blast furnaces was set up under the guidance of M. A. Pavlov in the Magnitogorsk, Kuznetsk, Makeevsk and other plants. The investigations were designed to answer many important questions concerning the planning, construction, and operation of blast furnaces of great power; empty doubts were expressed by some Soviet specialists as to the possibility of satisfactory work from blast furnaces of a wide type. The results of the study gave many new facts for the theory of the processes occurring in blast furnaces.

Pavlov initiated the study of the physical properties of blast furnace slag; beginning in 1932-1933 in many scientific institutions of the USSR.studies of the viscosity of the slag were carried out by Soviet scientists who were among the first in the world to do so. There was great value in the work of expanding the raw materials of the coke chemical industry which was carried out under the direction of Pavlov as chief specialist of the commission of the Academy of Sciences of the USSR. Thanks to this work the production of metallurgical coke was begun using coal which had not previously been utilized for coking. Pavlov also directed many other important scientific investigations in the Central Institute of Metals, in the laboratory for the metallurgy of cast iron of the Leningrad Polytechnic Institute, and elsewhere.

Even as a factory engineer Pavlov showed his ability as a planner and builder. During visits to the U.S.A. and to European countries – Sweden, Germany, France, Belgium, and England – he collected much material on the construction of shops and the building of furnaces. As a result of his critical evaluation of this material Pavlov

published the "Atlas of the Blast Furnace Industry," which passed through several editions. Later he also published "Atlas of the Open Hearth Industry." This was a valuable aid to constructors and students.

The great knowledge of Pavlov in this field permitted him to take an active part in planning activities which worked out the details of new metallurgical works. With his direct participation plans were made for blast furnaces with usable volumes of 930 and 1300 cu m.

Pavlov was always closely connected with industry. He often visited factories where he gave lectures and welcomed the plant engineers who came to him for advice. Although he was very busy, Pavlov always kept up a great correspondence and always found time to answer letters himself.

The activities of Pavlov as editor of the journals, Journal of the Russian Metallurgical Society, Soviet Metallurgy, and others, were very valuable in inculcating the most progressive methods in the industry of Russia and the USSR. He worked hard in improving the abstracts sections of these journals. M. A. Pavlov introduced the "critical abstract" in which the abstractor along with a summary of the essential points also gave his evaluation and critical remarks.

In his long professorial career Pavlov trained thousands of metallurgical engineers and scores of scientific workers. Many of his students are prominent scholars or noted engineers.

Pavlov has told of his life in detail in his famous book "Recollections of a Metallurgist." In this book he also gives interesting material on the development of the native metallurgy.

The Soviet Government highly honored the services of Pavlov, four times awarding him the Order of Lenin, as well as the Red Banner of Labor, and in 1945 naming him a Hero of Socialist Labor.

In spite of old age, Pavlov did not stop creative work. He continued to work on new projects, adding valuable contributions to Soviet science.

Soviet metallurgists will always keep in their hearts warm memories of Mikhail Aleksandrovich Pavlov – a remarkable scholar and patriot.

METALLURGY ABROAD

CONVERTER STEELMAKING IN AUSTRIA

Cand. Tech. Sci. S. G. Afanas'ev

TsNIIChM (Central Scientific Research Institute of Ferrous Metallurgy)

The delegation of Soviet metallurgists had the opportunity to see the largest metallurgical works in Austria: the Linz Works of the Fost Firm, the Works in Donawitz and the Erzberg mine, the high-grade steel works in Kapfenberg belonging to the Böller Firm and the works in Ternitz, Gonisberg and Mirzuschlag belonging to the Scheller-Blechman Firm.

In 1956 the output of pig iron in Austria was 1,650,000 tons, steel – 2,028,000 tons, and rolled product – 1,400,000 tons.

Most of the Austrian works are situated in Styria, a very important iron ore region of the country.

It is very difficult to find a suitable site for large works in the mountainous terrain of Styria; therefore the plants of some works are situated at a distance of 5-10 km from each other.

Austria has large iron ore resources; total reserves are estimated to be 370 million tons. The largest deposit of iron ore is the Erzberg mountain, 750 m high, situated near Eisenerz. It may be described as an Austrian Magnitna mountain, but the ore there has a lower iron content. The ore is mined on 30 terraces, 24 m high.

View of the Erzberg mountain and the town of Eisenerz.

All Austrian iron ores have a low iron content (30-33%), a high manganese content (2%), and little phosphorus and sulfur. The ore is transported by 20-ton dump trucks and electric cars from the mining places

to the beneficiation plant situated within the mining district. After the beneficiation, the iron content in the ore reaches 40-42%.

Austria has only two integrated iron and steel works with a full metallurgical cycle: the works in Donawitz which has four blast furnaces of 500 to 1100 cu m volume, and the works in Linz which has four blast furnaces of 850 and 1000 cu m volume.

At the Donawitz Works, pig iron of the following chemical composition is obtained from the Styria ores, %:

C	Mn	Si	P	S
4.0	2.2-2.7	0.1-0.3	0.010	0.05-0.07

According to Austrian experts, the silicon content in pig iron from the Donawitz Works is due to the composition of the raw materials employed and the peculiarities of the blast-furnace operation at that works.

Apart from Styria ore (60%), the blast-furnace plant at the Linz Works uses Swedish and Brazilian ores (20%) and pyrite cinders (15-20%); therefore the pig iron from this works contains 0.8% Si and about 2% Mn.

The desulfurization of pig iron with soda outside the blast furnace is extensively employed at both works.

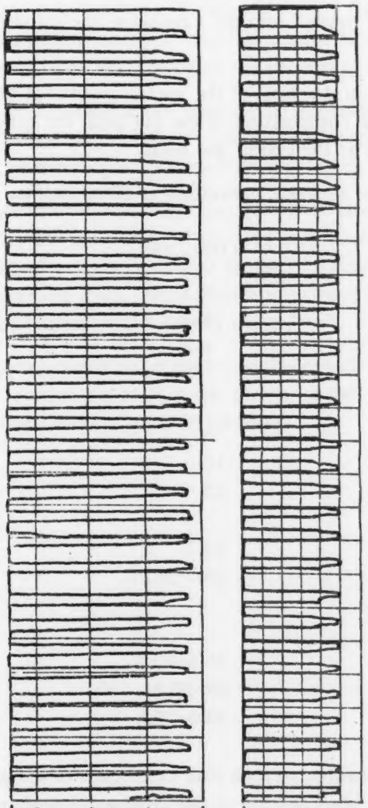


Fig. 1. Diagram of oxygen blowing into metal in converter (the works in Linz).

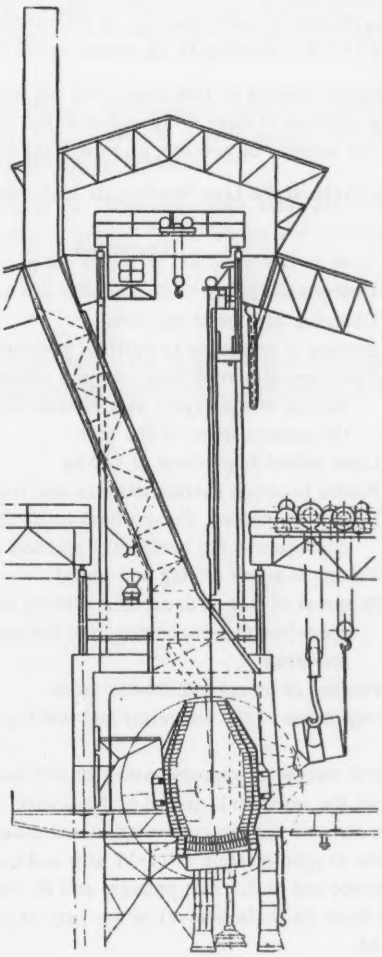


Fig. 2. Section through converter plant at the works in Donawitz.

In the field of steelmaking the Austrian steel industry is characterized by the trend toward the manufacture of high quality electro-steel. The manufacture of ordinary, as well as high quality steel in converters, using oxygen blown from above, is also characteristic of the Austrian steel industry. In 1956 more than 40% of all steel was made in converters with the use of oxygen.

The extensive development of this method is due to the high quality of the steel produced. The machine-building industry and other consumers who previously used exclusively steel made in the open-hearth furnace, are now willingly buying converter steel made with the use of oxygen.

A distinguishing feature of the technology of the process employed in converter plants in Austria is the application of afterblow (except at the Donawitz Works) and the addition of fluorspar into the charge for the fluidization of the slag. Only scrap is used as the cooler at both works. Lime, limestone, and fluorspar are used for slag separation.

A great deal of attention is paid at Austrian works to the quality of the lime — its constancy of composition and the size of particles. The silicon oxide content in the lime is approximately 1.5% and the losses on calcining ~7%. The Austrian experts consider that the duration of the slagless period in the converter process depends on the quality of lime and they correlate with this factor the amount of iron loss in burning,the ejection of metal, and the durability of the lining.

With the object of a better temperature control and the formation of a reactive slag, one part of the lime is added simultaneously with lowering of the tuyeres and the blowing of oxygen into the converter; the remaining part is added in a few portions in the course of the blast.

The charge consists of 18% scrap, 82% pig iron, and 5-7% lime (on the basis of the metal content of the charge). The amount of steel scrap added to the charge depends on the composition of the pig iron, the grade of steel made, the method of pouring, and the final temperature of metal at the end of the blast.

A heat cycle at the Linz Works lasts 32.5 minutes. The timing of separate operations is given below.

Operation	Time, min.
Charging of scrap	0-2
Charging of molten pig iron	2-4
Setting of converter in vertical position, addition of lime	4-5
Commencement of blast; oxygen pressure 10 atm; the nozzle of the tuyere at a distance of 900 mm above the surface level of the bath	5
Lime added in portions of 100 kg	13
Flame becomes shorter, darkens and begins to pulsate	23
Lifting the tuyere, discontinuation of the blast, setting the converter in the horizontal position	23.3
Taking samples of slag and metal	23.5-25.5
Removal of slag with rabbles, adding of lime and ground open-hearth furnace slag into the mouth of the converter	25.5-29.0
Pouring of metal into 30-ton ladle	29-30.5
Inspection of the converter and fettling of the throat	30.5-32.5

The laval nozzle of approximately 40 mm diameter is used for blowing oxygen into converters of about 35 ton capacity at the works in Linz and in Donawitz. The diagram of the blast regime at the Linz works is shown in Fig. 1. The curves of oxygen consumption and pressure indicate a regular blast regime. For the first 14 minutes of blowing, the oxygen pressure is 10-11 atm and consumption 110 cu m/min. For the subsequent 4 minutes the blowing is carried out at 8.5 atm pressure and 85 cu m/min oxygen input. The tuyere is situated at a distance of 800-900 mm from the surface level of the bath at rest, and its position remains unaltered throughout the whole blowing period.

The blast process is violent at the above-mentioned oxygen input, but it proceeds without large eruptions. The process is stopped, according to the form and color of the flame, at a low carbon content (0.03-0.05%).

For the determination of oxygen input, there is a meter with an integrater which allows, at any time during the operation, the determination of the amount of oxygen consumed from the commencement of the process. The work of the foreman conducting the operations is thus facilitated considerably.

Special attention is paid to the temperature of the metal: the temperature of rimmed steel on discharging is usually 1590-1620°C; that of the carbon steels, 1560-1680°C.

Steel is teemed from above into molds with hot tops, two ingots of 16 ton each being obtained. The diameter of the teeming nozzle is 80 mm.

The 24-hour output of the plant at the Linz Works, with two converters operating (the third converter is always kept in reserve or being overhauled), constitutes 2500-2600tons of steel; yield of useful materials on pouring from above is 88.4-88.7%. Oxygen consumption is 57 cu m per 1 ton of steel.

The technology of the steelmaking process at the Donawitz Works is similar to the one described above. As the composition of pig iron at the Donawitz Works is different from the composition of pig iron at the Linz Works (in manganese and silicon content), the consumption of coolers is somewhat lower.

The plant design at the Donawitz Works (Fig. 2) allows for the charging of scrap and liquid pig iron on one side and discharging of metal and slag on the opposite one.

It is significant that in spite of the difference in pig iron composition at these two works, the slags have approximately the same content of SiO_2. The amount of silica in the slag at the Donawitz Works is increased by the introduction of silica-containing additions (sand). In the opinion of Austrian experts, it accelerates the dissolution of lime and enhances the process of slag formation.

This method is not used in the converter plants of the Soviet Union. At home, in order to improve the conditions of slag formation when the silicon content in pig iron is low, bauxite is usually added, and the amount of FeO in the slag is increased by regulating the blast and varying the tuyere position.

As a result of a high manganese content in pig iron (2-2.5%) at the Donawitz Works, slags are quite fluid and after the blow most of the slag is discharged from the converter by gravity flow. Before the discharge of steel into the ladle, a barrier is made of ground open-hearth slag in the throat of the converter in order to retain the remaining slag. Such a method of steel and slag discharge is less effective than the discharge through taphole and spout in use in the Soviet Union.*

One's attention is drawn to the effective organization of production in the converter plants of Austrian works. During our stay in the plants we did not notice any stoppages in the production, all operations being carried out effectively and speedily.

* "Metallurgist" No. 3, 1956.

AUTOMATION OF THE BLOWER PLANT

The blower plant at our works contains four blowers of 60 cu m per minute compressed air output each, with asynchronous electric motors of 360 kw each and 6 kv voltage, and one electro-turbo-compressor of 200 cu m per minute compress air output, driven by an asynchronous motor of 1500 kw. In the same building there is 6 kv distributing equipment and a transformer 6/0.4 kv and 320 kw for local needs. The plant was previously operated by two electricians and two mechanics.

The section is now automated. Now, when the water pressure in the intermediate air coolers and the oil cooler falls, a loud bell – audible to the mechanic in any part of the blower plant – is sounded. The signal is also sounded when the temperature of the bearings of the turbo-blower or of the electric motor increases.

If the pressure of oil and water, which are delivered to the turbo-blower, falls below the minimum value allowed, the turbo-blower is automatically disconnected from the electric mains. Simultaneously an emergency oil pump is switched on, which pumps oil for the lubrication of the bearings and electro-turbo-blower decelerator during that time.

After introducing automation we were able to make one mechanic from each shift available for other work.

At the electric substation of the boiler plant there are three transformers 6/0.4 kv of 1000 kw, and 6 kv distributing equipment, the substation being operated by one man. After the adoption of automation and the arrangement for signalling to the boiler-plant electricians on duty regarding the condition of the transformers, the men on duty at the substation were made available for other work. The electrical equipment works quite reliably.

Automatic signalling of the oil level in the tanks of diesel motors and a regulator of the level in the turbine condenser has been installed on the steam-turbine blower (250 cu m per minute compressed air output) and on the diesel generators (1610 kw each). Now the auxiliary equipment of the steam turbine (condensate, curculation and draining pumps, and other equipment), of the steam-turbine blower and the diesel generators (air compressors, oil and water pumps, and oil tanks) is operated by one assistant mechanic per shift instead of two men, as was previously the case.

R. I. Baranov
"Krasnyi Oktiabr' " Works

QUESTIONS AND ANSWERS

A worker at the Frunze Metallurgical Works, N. G. Peteshenkov, in his letter to the editor of the "Metallurgist" asks the following questions, which arose in connection with the change-over to a 7-hour working day:

1. On working according to the scheme: 4 8-hr working days, followed by 48 hours off, the workers and shift personnel of the ITR work 48 hours overtime in a year. How should this work be compensated? Can these days be added to the annual leave?

2. How is the work done on public holidays compensated?

3. How are the average wages calculated?

We answer comrade Peteshenkov:

1. On the transfer of workers and the shift personnel of the ITR to the short working day, wages for the hours in excess of normal working hours are calculated in the month when the overtime work is evaluated. It is not permitted to add these days to the annual leave.

2. The work done on public holidays is paid at double rate. With the consent of the worker the monetary compensation can be substituted by a day off.

3. In the evaluation of wages for the holiday period and of compensation payment for holidays not taken, all forms of payments for work are taken into account including production bonuses, supplementary payment for overtime work, payment for lost time, payment for fulfilling of state and public obligations. The following payments are not taken into account in the evaluation of the mean wages: payments from the fund for promoting inventiveness; from the prize fund for socialist competition; bonuses from the fund of the works' director or the head of the plant, payments for occasional work which does not fall within the duties of the workers, etc.

ERRATA

In "Questions and Answers" in the "Metallurgist" No. 2, p. 40, the shift personnel of the ITR were erroneously included in the number of workers who are entitled to payments for overtime work. In accordance with paragraph 12 of the interpretation by the State Committee on Work and Wages, of the Council of Ministers of the USSR, supplementary payments to the managing and technical personnel for night work and for working time in excess of scheduled working time are included in their salaries.

METALLURGIST IN ENGLISH TRANSLATION

February, 1958

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SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosenergoizdat	State Power Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LEIIZhT	Leningrad Power Inst. of Railroad Engineering
LET	Leningrad Elec. Engr. School
LETI	Leningrad Electrotechnical Inst.
LETIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MEP	Ministry of Electrical Industry
MES	Ministry of Electrical Power Plants
MESEP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTI	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NII ZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOE	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIEL	Central Scientific Research Elec. Engr. Lab.
TsNIEL-MES	Central Scientific Research Elec. Engr. Lab.—Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIESKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZEI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us. — Publisher.